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PRELIMINARY LAYOUT SCHEDULE AND COST ESTIMATE FOR OIL SHALE SHAFT, SURFACE PLANT, HANDLING AND DISPOSAL FACILITIES. PICEANCE CREEK BASIN, RIO BLANCO, COLORADO.

PREPARED FOR:

UNITED STATES DEPARTMENT OF THE INTERIOR
BUREAU OF MINES

BY:

THE CEMENTATION COMPANY OF AMERICA, INCORPORATED.
2439 WEST WETMORE ROAD, TUCSON, ARIZONA 85705

Cementation

FINAL REPORT

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PRELIMINARY LAYOUT, SCHEDULE AND COST ESTIMATE

FOR

OIL SHALE SHAFT SURFACE PLANT
HANDLING AND DISPOSAL FACILITIES
PICEANCE CREEK BASIN RIO BLANCO COLORADO

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By

THE CEMENTATION COMPANY OF AMERICA INCORPORATED
2439 West Wetmore Road Tucson Arizona 85705

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The views and conclusions contained in this document are those of the authors and should not be interpreted as necessarily representing the official policies or recommendations of the Interior Department's Bureau of Mines or of the U. S. Government.

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16. Abstract

Provide preliminary layouts, cost estimates and time schedule for the construction of a circular 20 foot diameter concrete lined shaft to a depth of 2560 feet, included is a surface facility for loadout of 7500 TPD of oil shale based on a mining production time of 19.5 hours per day 3 shifts per day; 6 day week. Facilities for handling and/or stockpiling oil shale and waste rock are also included.

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U.S.B.M. OIL SHALE PILOT MINE

1. Introduction

The United States Bureau of Mines is carrying out a major research study to develop and demonstrate technology for mining the deep, thick saline rich oil shale deposits of the Parachute Creek Member of the Green River Formation in the Piceance Creek Basin of Colorado. The Cementation Company of America has been commissioned to prepare an engineering study on the shaft construction and surface plant facilities for a Pilot Mine operation in the region of Horse Draw. The purpose of this study is to develop order of magnitude estimates of construction costs and time schedules so that the economic feasibility of the project may be determined.

The study is based on the following conditions:

(i) The shaft is circular and 20 ft. in diameter, lined with concrete or reinforced concrete over the full depth.

(ii) The shaft is vertical and will be used for production and service.

(iii) The shaft is a downcast shaft.

(iv) The main production level will be at the bottom of the Saline Zone which in this area is at approximate elevation 3926 ft. There will be two other production levels within the 910 ft. thick Saline Zone (Lower Zone).

(v) The hoisting capacity, based on an available production time of 19.5 hours per day, 3 shifts per day and a 6 day week, will be approximately 7,500 tons per day.

(vi) The mine ventilation requirement is 500,000 cubic feet per minute.

The majority of the information regarding geological and hydrological conditions relating specifically to this area has been taken from coreholes U.S.B.M. 01-A and U.S.B.M. 02-A. Appendix I lists the information supplied

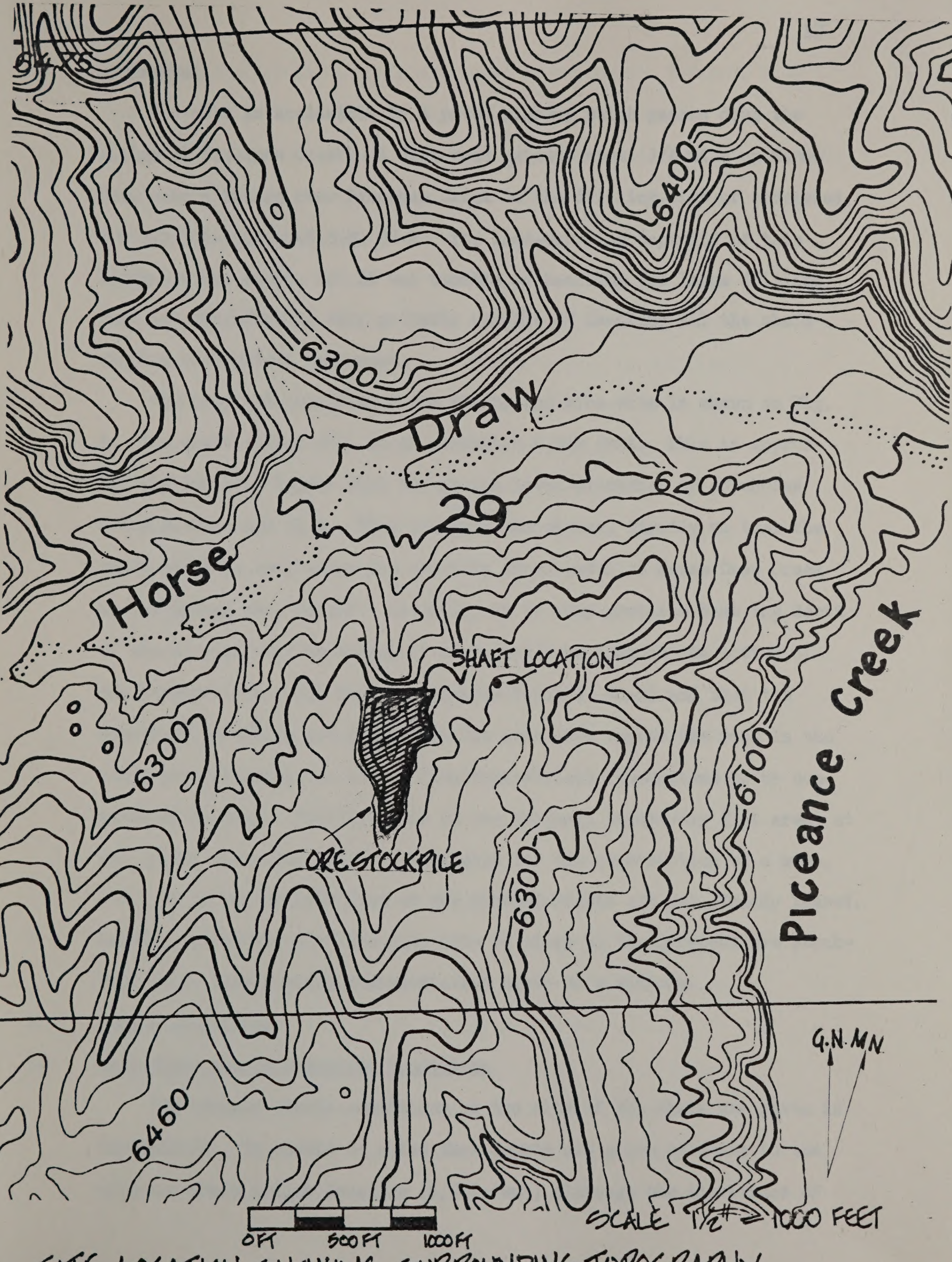
to the Cementation Company. Further information was gathered by personal observation at the core showing of U.S.B.M. 01-A in Laramie on January 21, 1976, and at the drilling site of U.S.B.M. 02-A and by communication with U.S.B.M. and U.S.G.S. personnel involved in the Oil Shale Project. More general background information on the region has been obtained by private communication and from the published data.

It is appreciated that further information relative to this study is being prepared at the present time, but at the time of preparation of this report, this data was not available. Therefore, the conclusions and recommendations made here have been based on that information that was available as listed in Appendix I. In particular, it has been necessary to make assumptions regarding the hydrological conditions existing in the area which play a significant role in determining the shaft sinking techniques and therefore costs. However, once more detailed information is made available, the assumptions can be reviewed and techniques and estimates amended if and as required.

2. Topography and Surface Location

The lower regions of Horse Draw are to be found in Section 29, of Township 1S and Range 97W. See contour map, Fig. 1. This is located towards the center of the plateau which comprises the Piceance Creek region of Colorado. In general, the land surface slopes gently downwards from the rim towards the central part of the plateau. Most of this central area is characterized by rough rolling hills. This is in contrast to the Southern rim which is deeply dissected and terminates in a very irregular line of spectacular precipitous cliffs. Elsewhere the rim of the plateau is more regular and less precipitous. The underlying strata also generally slopes from the rim to the center and so, at Horse Draw, being at the center of the basin, the deposits are mainly

FIGURE 1



SITE LOCATION SHOWING SURROUNDING TOPOGRAPHY

horizontal.

The area is accessible by a paved highway which passes down the Valley of Piceance Creek. A dirt road extends about 3/4 mile up Horse Draw over a bridge over Piceance Creek to the drilling site of coreholes U.S.B.M. 01-A and U.S.B.M. 02-A. The load-carrying capacity of this bridge is not known, but it was capable of handling the large drilling rigs and therefore is very probably capable of handling all the shaft sinking contractors equipment.

The proposed location of the shafts and mine site is shown on Fig. 1. The shaft collar will be at elevation 6,340 feet. This is approximately 100 feet higher than the ground level elevation of coreholes U.S.B.M. 01-A and 02-A. This location was chosen, firstly so that the shaft would be well above the possible flood level of Horse Draw Creek in the Spring snow runoff. Secondly, it is very probable that the depth of the alluvium on the slopes is very much less than in the bottom of the valleys (viz ± 140 feet in corehole U.S.B.M. 01-A) and that the necessary levelling and grading of the site will expose the rock in the shaft area allowing the collar (headframe/hoist) foundations to be constructed on rock. Thirdly, this is the largest, relatively flat area, at the lowest elevation which is available for the construction of a mine, i.e., it is not private land or the mineral rights are not already leased. Lastly, the location of the mine site is close to the proposed ore stockpile area, thus keeping transportation costs to a minimum.

3. Strata Conditions

3.1 Geological and Hydrological Conditions

The assumed strata conditions at the site of the shaft are given in Appendix II. It should be noted that depths are given relative to the proposed shaft collar location (6,340 feet) although the elevations of

different geological horizons are taken from corehole U.S.B.M. 01-A. Appendix III is the report by Cementation Research Limited of their study of the available geological and hydrological information. Depths are given relative to the ground level elevation at the collar of the corehole being referred to.

3.2 Ground Treatment

The hydrological situation is stated in the summary of the report from Cementation Research Limited (Appendix III) as follows:

> "To reach the ore deposits the shaft will have to pass through two major aquifers before entering competent rock at a depth of some 1500 feet. Apart from a 100 feet thick 'barrier' zone at about 950-1050 feet depth the entire formation above 1500 feet must be regarded as potentially wet. We estimate a total water inflow to a 28 feet diameter excavation of between 10,000 and 20,000 gallons/minute, with the main inflow zones 500-900 feet and 1050-1400 feet. The rock in these zones is generally weak and highly fractured and in the lower aquifer contains frequent collapsed "rubble zones".
>
> To sink a 'dry' shaft ground treatment will be required from about 200 feet to a depth of about 1500 feet. On the evidence available, and subject to a number of conditions, we believe that freezing is a more feasible means of ground treatment than grouting in the present instance. Grouting would necessarily be very extensive and somewhat unpredictable especially since ground strengthening as well as water stopping will probably be required

in certain zones. A far more detailed "picture" of the formation is needed to design such an exercise. In-situ freezing and/or grouting trials are recommended prior to shaft sinking."

The conditions which should be met prior to sinking the shaft using freezing techniques are outlined on Page 10 of Cementation Research Limited's report. The major problem which could influence the choice of freezing as the more satisfactory process of ground treatment is given as follows: "It is generally considered that a ground water flow rate of some 3-5 feet/day is the maximum that can be tolerated for a successful freeze operation. More knowledge....is required before we can be certain on whether ground water flow rate is likely to be a problem."

The information that is available, though not sufficiently detailed to be able to make conclusive statements does indicate that this condition is satisfied and further treatment such as pre-freeze grouting will not be necessary.

The time and cost estimates for shaft construction have therefore been based on the freezing method being used to control the ground water during sinking.

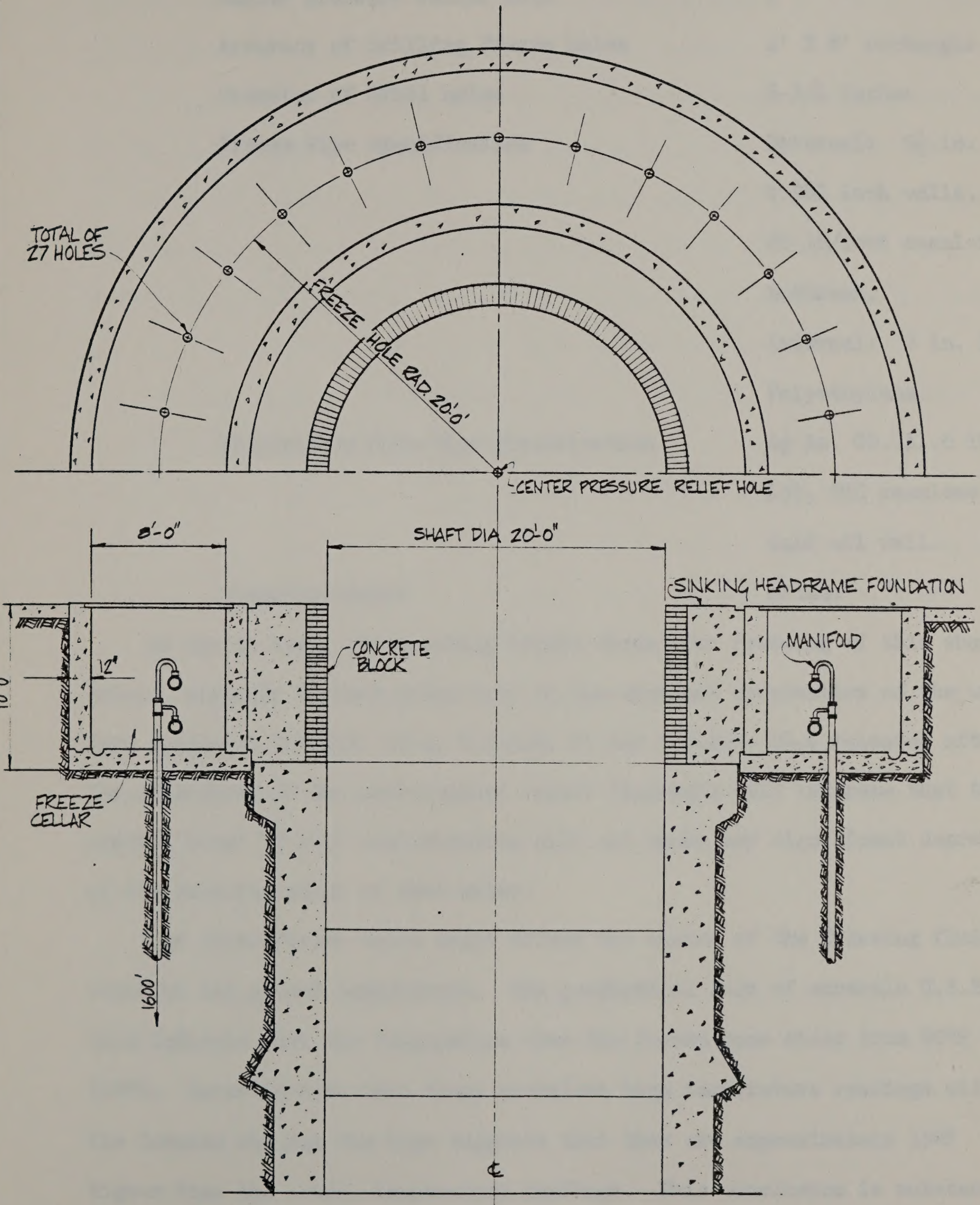
3.2.1 Freeze Layout

The arrangement of the freeze holes around the 20 foot diameter shaft is shown in Fig. 2.

The specifications are given below:

Diameter of freeze hole circle	40 ft.
Number of freeze holes	27
Distance between freeze holes	4.65 feet
Number of observation holes	2

FIGURE 2



LAYOUT OF FREEZE HOLES

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Depth of freeze holes	1600 feet
Center pressure relief hole	1
Accuracy of drilling freeze holes	4' X 8' rectangle
Diameter of drill holes	8-3/4 inches
Freeze Pipe specification	External: 5½ in. OD. 0.361 inch walls, 20 lb/foot seamless buttress. Internal: 2 in. ID. Polyethylene.
Observation Hole Pipe Specification	4½ in. OD. 11.6 lb/foot, K55, STC seamless standard oil well.
Freezing Liquid	Brine.

As can be seen, the freezing liquid chosen for freezing of this shaft is brine. Analysis of some prime data on the chemical composition of the water from coreholes U.S.B.M. 01-A, U.S.B.M. 01 and U.S.B.M. 02-A released after the publishing of the hydrological report (Appendix III) indicate that the overall level of salt concentration will not cause any significant depression of the freezing point of that water.

One other factor which could effect the choice of the freezing fluid would be the ground temperature. The geophysical logs of corehole U.S.B.M. 01-A indicate that the temperature over the frozen zone rises from 90°F to 110°F. Investigation into these anomalous high temperature readings with the Company who ran the logs suggests that they are approximately 35°F higher than the actual temperature readings. This conclusion is substantiated by readings from corehole U.S.B.M. 02-A which are approximately 35°F

lower than those from U.S.B.M. 01-A. For example, at 1500 feet, at the bottom of the frozen zone, the temperature in U.S.B.M. 02-A was recorded at 71°F. In conclusion, therefore, it is not considered necessary to use more sophisticated freezing liquids than the conventional brine.

It will be noted that a pressure relief hole is required down the center of the shaft. This relieves the pressure which builds up in the formation as the icewall extends inwards. As indicated earlier, more detailed geological and hydrological data will be required prior to sinking the shaft. Preferably one hole should be drilled down the center of the shaft. Following testing etc., this hole can be used as the pressure relief hole.

On completion of sinking and lining to 1560 feet, the freeze plant will be closed down and, depending upon the remainder of the shaft sinking schedule, it may be necessary to circulate heated brine through the freeze holes. This is done to ensure that the icewall has thawed so that back lining injection can be carried out prior to completion of the shaft sinking phase (see lining details page 19).

On the completion of this thawing process, the freeze holes are abandoned by taking out the freeze pipes and carefully filling the holes with cement grout. This eliminates the danger from a leaking freeze hole at a later date and stops the migration of water between aquifers.

4. Shaft Design

4.1 Legal Requirements

In addition to other factors the legal requirements must be considered. In Colorado, these are defined in the "Colorado Bureau of Mines, Mining Laws." The sections that are quoted below are those which are relevant to the design of the shaft under consideration.

Section 57, "Escapeways and Exits" states:

1. Any underground mine, quarry or other excavation opened

by a shaft which is vertical, inclined or a combination of both and which reaches a vertical depth of one hundred (100) feet, shall have two (2) openings or exits to the surface, that is one exit in addition to the shaft mentioned.

5. The surface outlet for any of the above exits shall be not less than one hundred (100) feet apart.

7. When there are two (2) exits from the underground to the surface, the employer shall provide, in addition to any mechanical means of ingress or egress, at least one (1) means of outlet for employees with ladders from the lowest workings of the mine to the surface.

9. In all developed and operating mines, it shall be the duty of the employer to maintain and keep in good condition and repair at least two (2) outlets to the surface or there shall be an underground connecting passageway between every such mine and some neighbouring mine so that in any case there shall always be at least two (2) distinct and available means of access to the surface for all employees underground.

10. When new mines are opened or new levels opened in an old mine, the employer shall be allowed a reasonable time in which to secure a second exit or escapeway, but not more than six (6) months from the time such opening is made. Not more than ten (10) persons may be employed in that part of the mine or level until such escapeway or exit is made.

11. Under any condition where it is obviously impracticable for an employer to have two (2) escapeways or exits in any part of an underground working operations may be allowed with only one (1) outlet and under such conditions and with such precautions as the Commissioner of Mines may prescribe and then only upon written permission.

Section 140 "Shafts and Raises" states:

1. All shafts or raises equipped with hoisting machinery, which are used to handle men and materials, shall be divided into at least two (2) compartments, one of which shall be partitioned and set aside for a manway.

Section 145 "Skips" states:

4. Skips, used for hoisting or lowering employees or other persons in any vertical shaft or any in line shaft, shall be covered with an iron bonnet to protect employees from falling rock. They shall travel in guides or on rails with back runners to prevent derailment and shall have safety catches that will hold the fully loaded skip, should the hoisting rope fail.

A similar paragraph is stated in Section 23 'Cages' Article 3.

To summarize, therefore, the legal requirements are that two shafts will be required at the proposed mine unless permission to operate with one shaft is obtained. The service shaft shall have at least two compartments, one of which is a manway, and the service shaft shall be equipped with rigid guides and support sets.

The alternative arrangements have been considered assuming that permission could be obtained for operating with one shaft and with rope

guides. With the single shaft system, a divider would be installed to provide the separate intake and return ventilation compartments. The main shaft conveyances would be installed in one compartment, say the intake, and the return compartment would be unfurnished. However, this arrangement in a 20 foot diameter shaft would not be able to handle the ventilation requirements (see next section), nor would the layout of skip and cage in the shaft be a practical arrangement. In this study it is therefore assumed that at least one other shaft already exists which constitutes the second exit to the surface and is the upcast shaft capable of handling 500,000 cubic feet of ventilating air per minute.

The use of rope guides in the two shaft system has also been considered, but to obtain the necessary clearances between conveyances severely limits the size of the conveyances. The use of rope guides therefore has been discarded because as much space should be reserved for the movement of heavy, irregular shaped equipment in this service shaft.

4.2 Ventilation

The shaft cross sectional area and the area of the shaft conveyances, together with the type of furnishing in the shaft are the limiting factors in determining the quantity of ventilating air that can be reasonably handled in the shaft. Generally accepted criteria are:

(a) The area of the shaft conveyances should not exceed 50% of the shaft area.

(b) Limiting air velocities:

In a furnished shaft with rigid guides and support sets	1500 ft/min
In a furnished shaft with rope guides	2000 ft/min
In an unfurnished shaft	2500 ft/min

In the shaft under consideration therefore, the limiting air velocity can be taken as 1500 ft/minute. This means that in this 20 foot diameter shaft, the total quantity of ventilating air that can be readily handled by the shaft without having excessive friction loss, high water gauges and power consumptions is 471,000 c.f.m. Strictly speaking, a 20 foot diameter shaft is too small for the ventilation requirement of 500,000 c.f.m. and a 22 foot diameter shaft would be more appropriate. However, the discrepancy is small between the ventilation requirement and that which can be readily obtained in this 20 foot diameter shaft and there is no doubt that this sized shaft will be adequate.

4.3 Shaft Lining

Shaft linings in water bearing strata must, for the life of the shaft, be able to withstand the pressures which will be applied and keep the residual shaft water inflows within acceptable limits. In choosing the type of lining which will best provide these properties, for each shaft diameter, consideration must be given to:

(a) The type and magnitude of loading on the lining. For the same pressure, loads on a lining and therefore lining strengths increase with increased shaft diameters.

(b) The water sealing properties of the lining type. For example with a concrete lining the difficulty and probability of completely sealing construction joints decreases with increased hydrostatic pressure. This is less of a problem with either

steel or cast iron linings.

(c) The nature and distribution of unconsolidated deposits. For example, where a lining covers unconsolidated deposits, such as running sand, and there is leakage through the lining, then it is likely that over a period of time the leakage will increase due to erosion of the lining by sand particles carried in the water, and cavities will be created behind the lining due to the removal of the sand. To correct such a condition, extensive and costly repair work would be required during the operating life of the mine.

There are three types of lining which will be considered for these shafts.

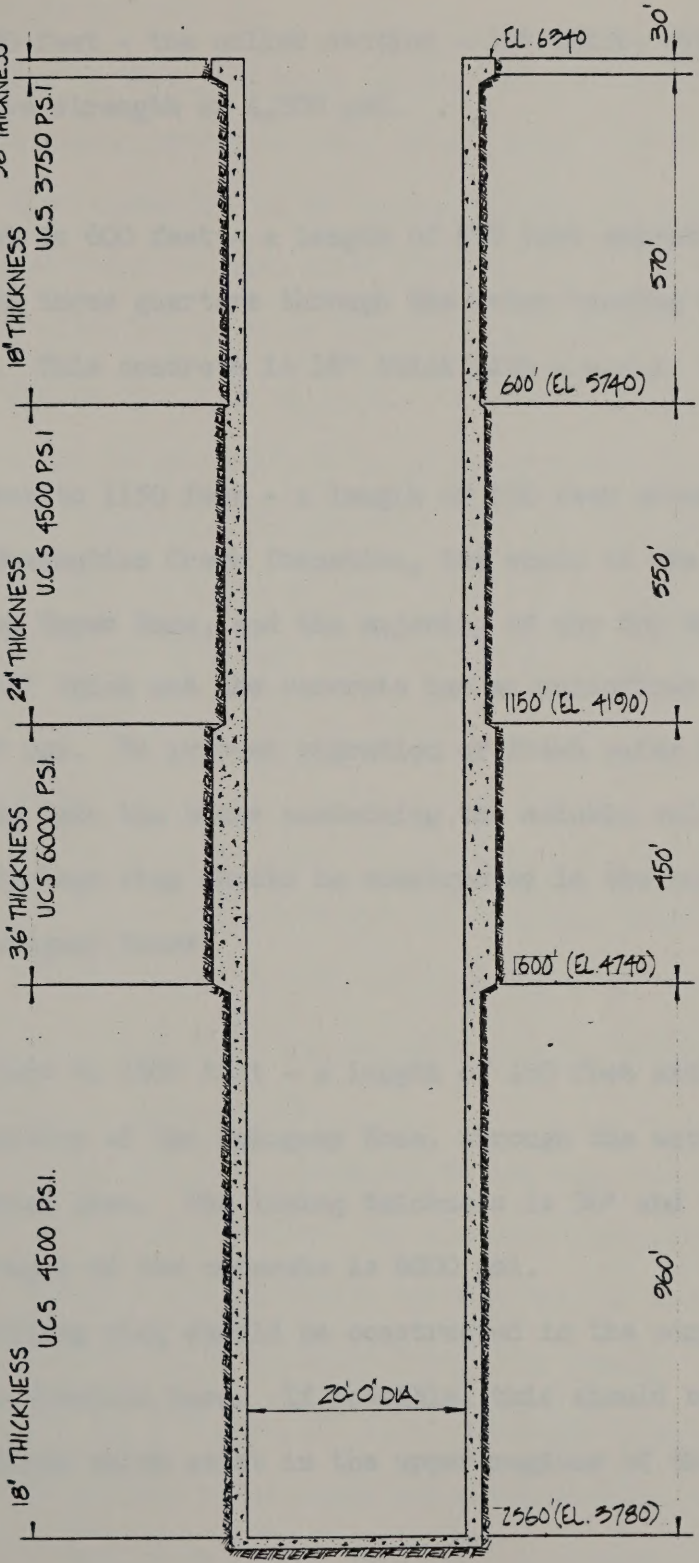
(a) Concrete or reinforced concrete.

(b) A combination of steel liner plate and concrete.

(c) A combination of cast iron tubbing and conrete.

In the majority of circular shafts, concrete linings are used since in anything but exceptional circumstances this type provides the strength and sealing properties required. Although there will be unconsolidated sections of strata in this section, these will not be of a 'sandy' nature and material removal by leaking water need not be considered.

The lining details for the shaft are shown in Figure 3. Concrete has been specified throughout. The introduction of rebar into the concrete lining of frozen shafts causes damage to the concrete when the freeze is taken off. This is caused by the differential contraction between the concrete and the steel. Furthermore, provision should be made for cutting any steel which is continuous through the concrete which may be necessary to install for contruction, i.e., hanging rods. The lining thicknesses



LINING DETAILS

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and the concrete strengths in the various sections are as follows:

Section 1

From 0 to 30 feet - the collar section - 36" thick, with an unconfirmed compressive strength of 4,500 psi.

Section 2

From 30 feet to 600 feet - a length of 570 feet extending from the collar section to three quarters through the water bearing Evacuation Creek formation. This concrete is 18" thick with a u.c.s. 3750 psi.

Section 3

From 600 feet to 1150 feet - a length of 550 feet covering the lower quarter of the Evacuation Creek formation, the whole of the upper part of the water bearing Upper Zone, and the majority of the dry Mahogany Zone. This lining is 24" thick and the concrete has an unconfined compressive strength of 4500 psi. To prevent migration of fresh water downwards behind the lining into the zones containing the soluble salts, a seal consisting of Pikotage ring should be constructed in the competent dry ground of the Mahogany Zones.

Section 4

From 1150 feet to 1600 feet - a length of 450 feet extending from just above the bottom of the Mahogany Zone, through the water bearing incompetent Leached Zone. The lining thickness is 36" and the unconfined compressive strength of the concrete is 6000 psi.

A second sealing ring should be constructed in the competent dry ground below the fracture base. If possible, this should be above the soluble Halide Zones which exist in the upper regions of the Lower Zones.

Section 5

From 1600 feet to the shaft bottom through the remainder of the lower

Zone and into the top of the Garden Gulch, which are both dry and competent, the lining thickness is 18" and the concrete has an unconfined compressive strength of 4500 psi.

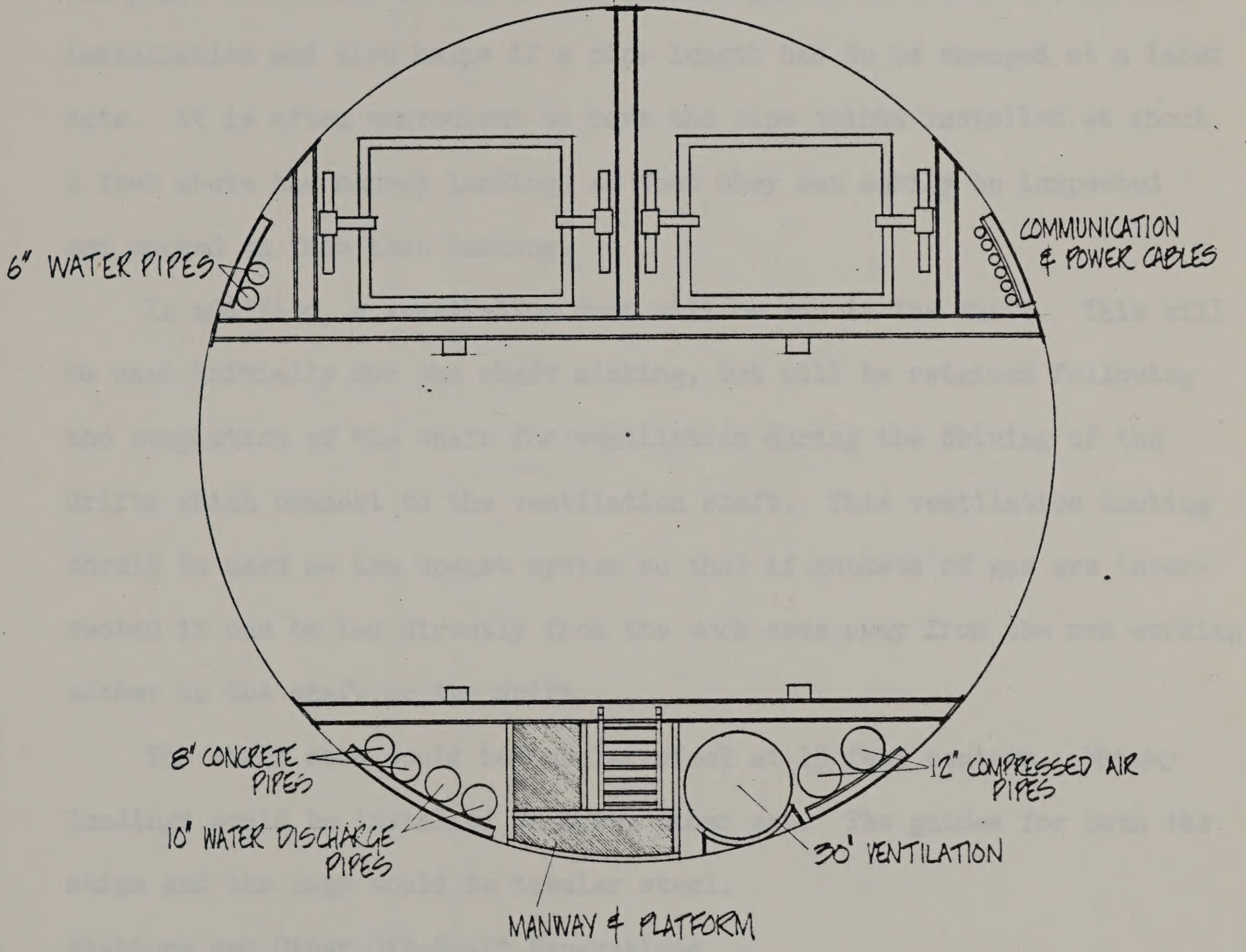
It is anticipated that the shaft lining will be constructed in a series of 20 foot pours (probably less, when lining through the incompetent ground 120 feet above the water table). Each pour will therefore have a construction joint through which there will be, without doubt, a relatively easy path for water to flow into the shaft. However, this water can be minimized by using correctly designed construction joints using water stops and by systematically grouting behind the shaft lining on the completion of sinking. In this way, the residual water flow into the shaft could be kept to below 30 gpm. This water would be controlled and collected by a series of water rings and drain pipes which direct the water into the permanent mine pumping system.

4.4 Internal Shaft Layout

The layout of dividers and shaft furnishing etc., has been developed in conjunction with the hoisting systems and the requirements of the shaft in terms of ore hoisting capacity and men/materials/equipment handling. The details of the shaft conveyances are given in the section dealing with the hoist and hoisting arrangements.

The shaft contains two skip compartments and one cage compartment, see Fig. 4. The main divider between the two skip compartments and the cage compartment supports the partition between the two sections. Further screens are supported by other dividers around the manway. The shaft also contains 8" concrete slick lines, 10" water discharge lines, 12" compressed air lines, 6" water supply lines, power and communication cables. The number of service lines of each type that will be required will depend

FIGURE 4



GENERAL ARRANGEMENT OF SHAFT SECTION

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very much upon the mining method used in the mine. Allowance is being made for installing two pipes of each type and for supplying 2000 kva of power underground. The pipe joints should be designed to make allowance for expansion of the pipe when the freeze is taken off the shaft. The pipes should all be cut to the same length as this aids the initial installation and also helps if a pipe length has to be changed at a later date. It is often convenient to have the pipe joints installed at about 4 feet above the manway landings as that they can easily be inspected and worked on from that landing.

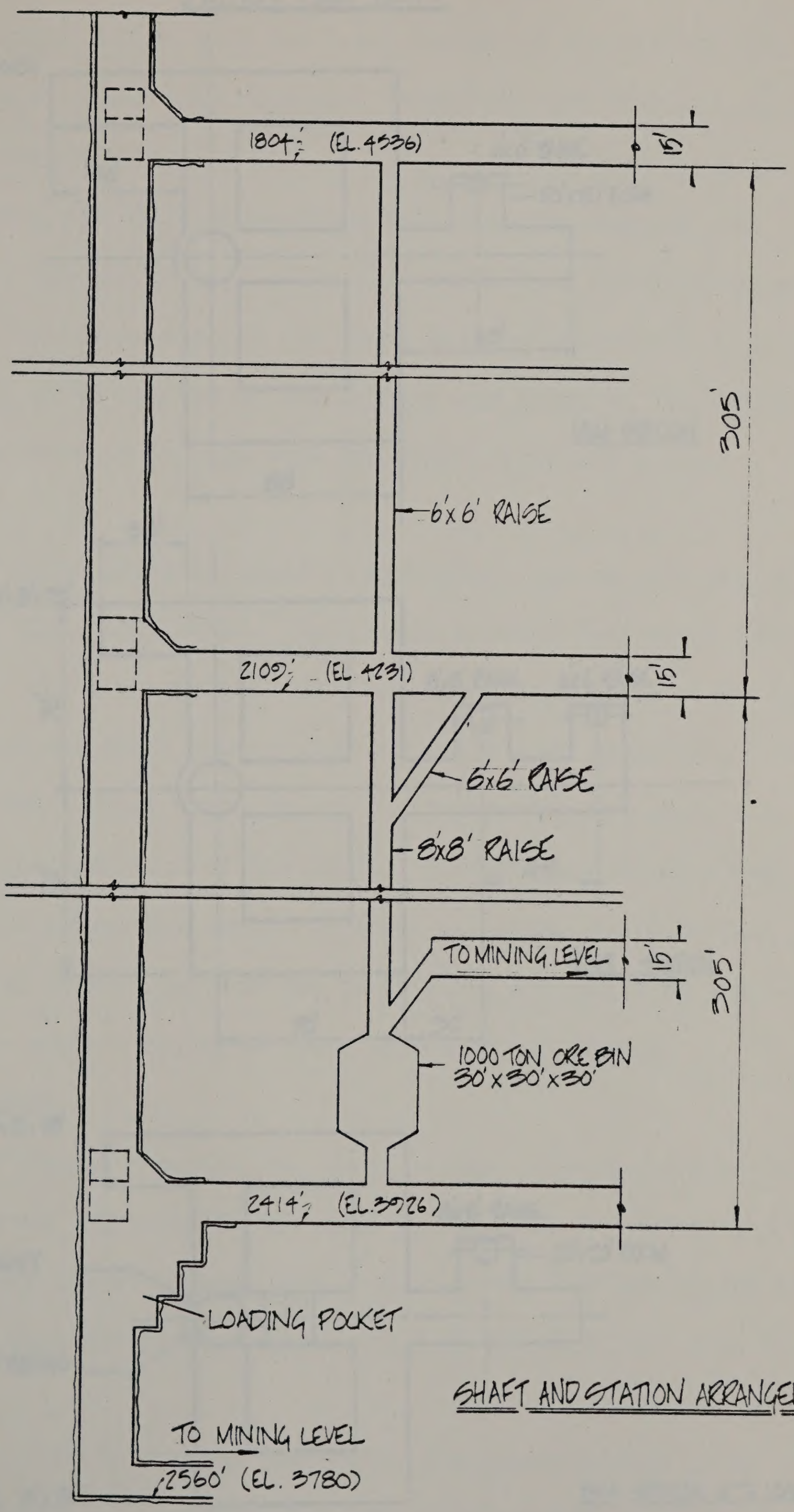
In addition, a ventilation duct will be run in the shaft. This will be used initially for the shaft sinking, but will be retained following the completion of the shaft for ventilation during the driving of the drifts which connect to the ventilation shaft. This ventilation ducting should be used as the upcast system so that if pockets of gas are intersected it can be led directly from the work area away from the men working either in the shaft or the drift.

The shaft sets would be tubular steel at 10 feet centers. Manway landings would be installed on every other set. The guides for both the skips and the cage would be tubular steel.

5. Stations and Other Off-Shaft Excavations

The time and cost estimates are based on a general arrangement for shaft stations and other excavations as shown in Fig. 5a and Fig. 5b, and shows the extent of the work which should be done in conjunction with the shaft sinking. The layout is based on the assumption that all these levels will be developed simultaneously and all the ore is handled through one loading pocket at the lower level.

FIGURE 5a



SHAFT AND STATION ARRANGEMENT

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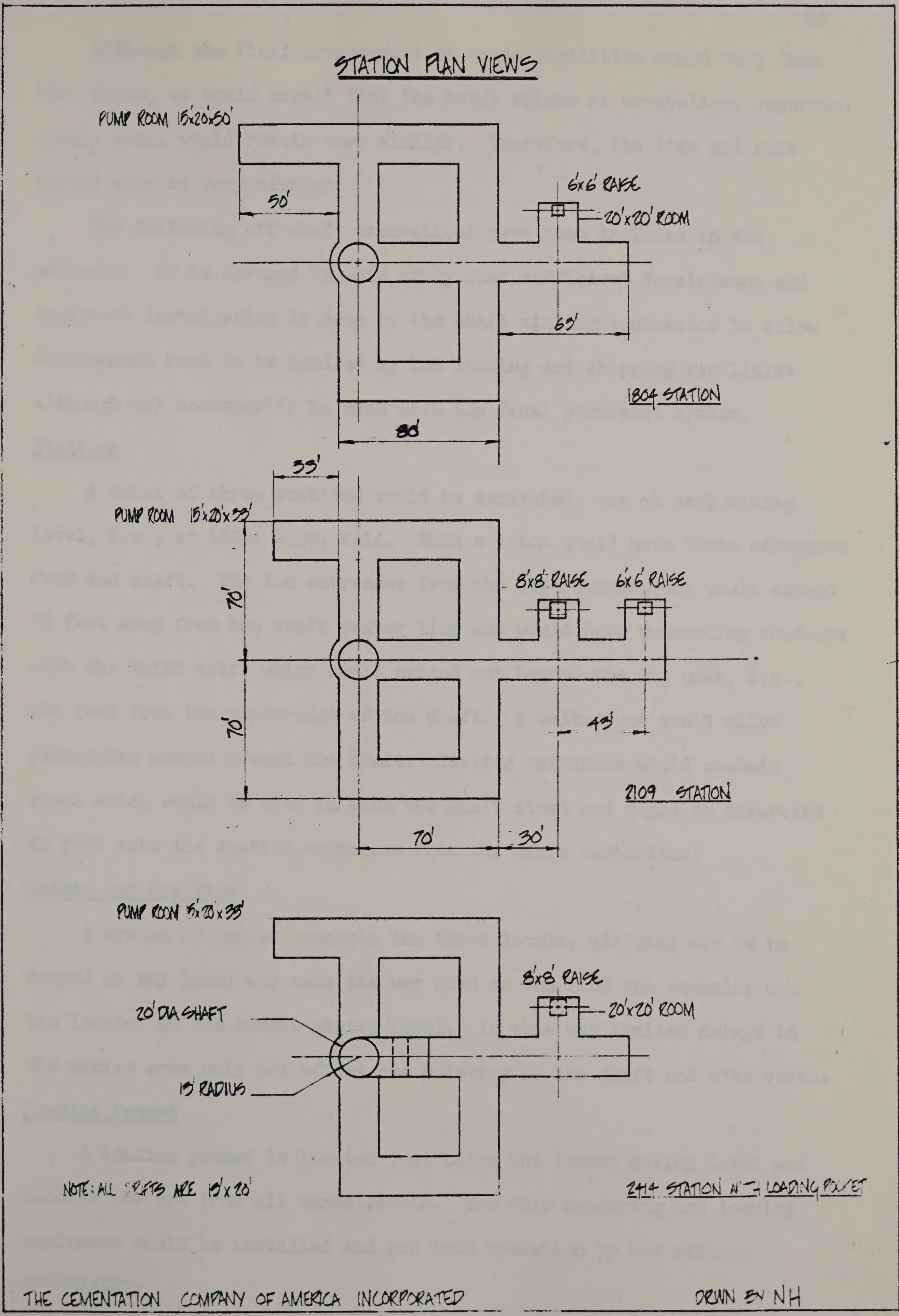
FIGURE 5b


STATION PLAN VIEWS
PUMP ROOM 15'x20'x50'
50'
6'x6' RAISE
20'x20' ROOM
65'
1804 STATION
80'
33'
PUMP ROOM 15'x20'x33'
8'x8' RAISE
6'x6' RAISE
70'
70'
43'
2109 STATION
70'
30'
PUMP ROOM 15'x20'x33'
8'x8' RAISE
20' DIA SHAFT
20'x20' ROOM
15' RADIUS
NOTE: ALL DRIFTS ARE 15'x20'
2414 STATION WITH LOADING POCKET
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Although the final arrangement of these facilities could vary from that shown, we would expect that the total volume of excavation, concrete, steel, etc., would remain very similar. Therefore, the time and cost should also be very similar.

The following off-shaft excavations have been included in the estimate. It is assumed in this study that sufficient development and equipment installation is done by the shaft sinking contractor to allow development rock to be handled by the loading and skipping facilities although not necessarily in each case the final permanent system.

Stations

A total of three stations would be excavated; one at each mining level, i.e., at 1804, 2109, 2414. Each station would have three entrances from the shaft. The two entrances from the cage compartment would extend 70 feet away from the shaft center line and would have connecting roadways with the third drift which would extend out beyond the ore pass, i.e., 100 feet from the centerline of the shaft. A walkaround would allow pedestrian access around the shaft. Station entrances would contain steel which would be tied in with the shaft steel and would be concreted 25 feet into the station entrances from the shaft centerline.

Raises and Ore Bins

A system of raises connects the three levels, allowing ore to be dumped at any level and make its way down to the 1000 ton capacity ore bin located at the lowest mining level. In this way limited delays in the mining area will not affect ore hoisting in the shaft and vice versa.

Loading Pocket

A loading pocket is located just below the lowest mining level and handles the ore from all three levels. The skip measuring and loading equipment would be installed and put into operation by the sinking contractor.

Pump Station

The main pump station would be located on the upper mining level and would be installed and brought into operation with the permanent pumps by the sinking contractor when this station is excavated as the shaft goes down. This pump station would be capable of handling the full head to the surface, and would be used for the remainder of the shaft sinking as well as for the life of the Pilot Mine. Other secondary pump stations would be located at each of the other two levels, but would use smaller pumps operating at lower heads which would feed the main pump station. Similarly, the secondary pump stations would be installed and brought into operation during the sinking and would handle the water discharge during the sinking below that level. Provision has been made in the estimate for supplying and installing pumping equipment capable of handling 500 gpm from the mine.

Further excavation is required in the general area of the shaft prior to the mine going into production. However, this can be carried out more economically together with the general mine development work on completion of the shaft work. Items in the category include for example, (a) the spillage cleanup slope; a decline driven from the lowest mining level to the bottom of the shaft to allow standard mine equipment to clean up any ore spillage in the shaft, and (b) the ore bin access; a roadway providing access to the top of the ore bin from the lowest mining level (c) permanent sumps.

6. Permanent Facilities

It is beyond the scope of this study to give a final and detailed specification for the permanent facilities system. There is a wide range of choice for each of the elements which make up a system and the

final specification must be the subject of an indepth study. Nevertheless, it is possible at this point to define a system which is feasible and by doing so arrive at a practical construction time and cost estimate.

6.1 Hoists

6.1.1 Alternative Systems

The production rate for the Pilot Mine is specified as 7500 tons/day, based on 19.5 hours of hoisting over 3 shifts. The density of the oil shale in place has been taken as 1.95 tons/cu. yd. (equivalent to an oil yield of ±20 gals/ton) and has a broken density of 1.21 tons/cu. yd. assuming 38% voids.

However, in addition to hoisting the oil shale, the shaft will be used to handle various types of mining machinery which will be tested in the Pilot Mine operation. There has been a dramatic increase in the physical size and weight of underground mining machinery in recent years. Some examples are given below and there are many others which could be equally suitable.

Shuttle Cars - up to 30 feet long, 12 feet wide and 6'2" high and weighing 60,000 lbs.

L.H.D.'s Wagner ST8 - up to 32'9" long, 8'0½" wide and 6'2" high. When broken at the hinge pin, the heaviest section is the chassis. This is 16'7" long, 8'0½" wide and weighs approximately 30,000 lbs.

Continuous Miners - the heaviest single component of the continuous miner is the gear box. A typical gear box weighs 65,000 lbs. without gears and 100,000 lbs. when the gears are in place.

In the absence of any specific knowledge of what equipment is required to go down the shaft, a maximum load capacity of 45 tons has been chosen for the service hoist. The service cage compartment is as large as other considerations will allow.

Several different hoisting systems have been studied for the Pilot Mine. They are as follows:

(a) Alternative 1. - Two Drum Hoists.

Production Hoist - a 144" Diameter X 84" or 60" face double drum hoist, handling two-12 ton capacity skips in balance.

Service Hoist - - a 144" diameter X 84" face single drum hoist, that can handle a normal load of 15 tons in the cage, or loads of up to 45 tons when slung below a double down sheave in the shaft.

This hoisting system would be flexible and it makes efficient use of the shaft cross-sectioned area.

(b) Alternative 2. - 2 Koepe Hoists.

Production Hoist - a 138" diameter four rope hoist handling a single 24 ton capacity skip and a counter-weight.

Service Hoist - a 150" diameter four rope hoist handling a single large skip and counterweight. Maximum cage load would be 45 tons.

This hoisting system would be flexible but inefficient. A 45 ton load would only be raised or lowered occasionally, but in order to cope with this load, the service hoist would have to be larger than the production hoist. In addition, the floor area of the cage is limited by the space required in the shaft for the two counterweight compartments.

(c) Alternative 3. - One Koepe and One Drum Hoist.

Production/Service Hoist - a 150" diameter four rope Koepe hoist handling a single 30 ton capacity skip, counterbalanced by a large cage. This hoist can produce the required 7500 tons of ore in just under 16 hours and be used to raise and lower major material loads for the balance of each day. As in the two previous cases, the maximum material load will be 45 tons.

Service Hoist - a 72" diameter X 54" face single drum hoist that will be used to transport supervisory staff and maintenance crews and their equipment.

This hoisting system would be efficient in terms of shaft layout and equipment utilization, providing shaft production and material handling schedules are carefully planned and rigidly adhered to.

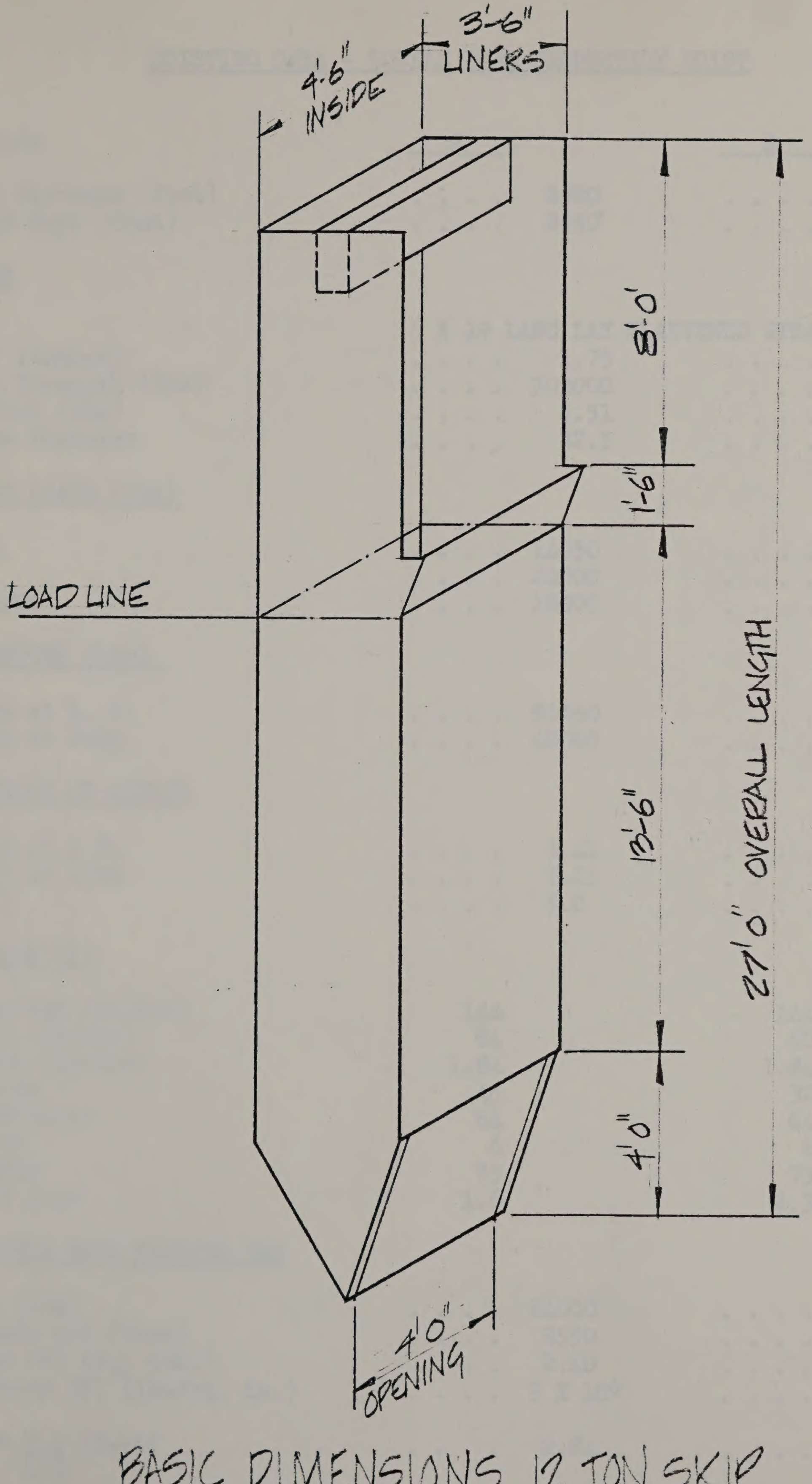
The prime function of this mine, being essentially an experimental mine, is not solely production. Of equal, if not more importance, is the testing of underground methods and equipment in the oil shales. It is therefore important that adequate facilities are provided for the transport of bulky and heavy items of equipment. With these points in mind, the overall hoisting system should have the maximum flexibility which is commensurate with both the largest cage cross section area available and load carrying characteristics. These conditions are satisfied most completely by Alternative 1 - two drum hoists. Although the purchase price of the two drum hoists is higher than either of the other alternatives, the extra cost is offset by the higher cost of the tower headframe required by the other alternatives. Taking every item into account, i.e., hoist, headframe, hoisthouse, foundations, hoist installation, skips, cages, ropes, sheavewheels, etc., the overall cost of alternative 1 is very similar to alternative 3. Alternative 2 is higher.

6.1.2 Double Drum Production Hoist

A skip load of 12 tons and a maximum speed of 1800 feet/minute has been selected as a basis for this hoist in order to meet the production requirements. The principal dimensions of the skip are shown in Fig. 6.

Details of the ropes, loads and drum parameters are given in Table 1. Two alternative arrangements have been considered. Alternative A has the two skips on opposite sides of the shaft and the central symmetrical portion, 7'8" between guides, across the widest part of the shaft, is the cage compartment, see Fig. 7. Alternative B has the two skips side by side and a service

FIGURE 6



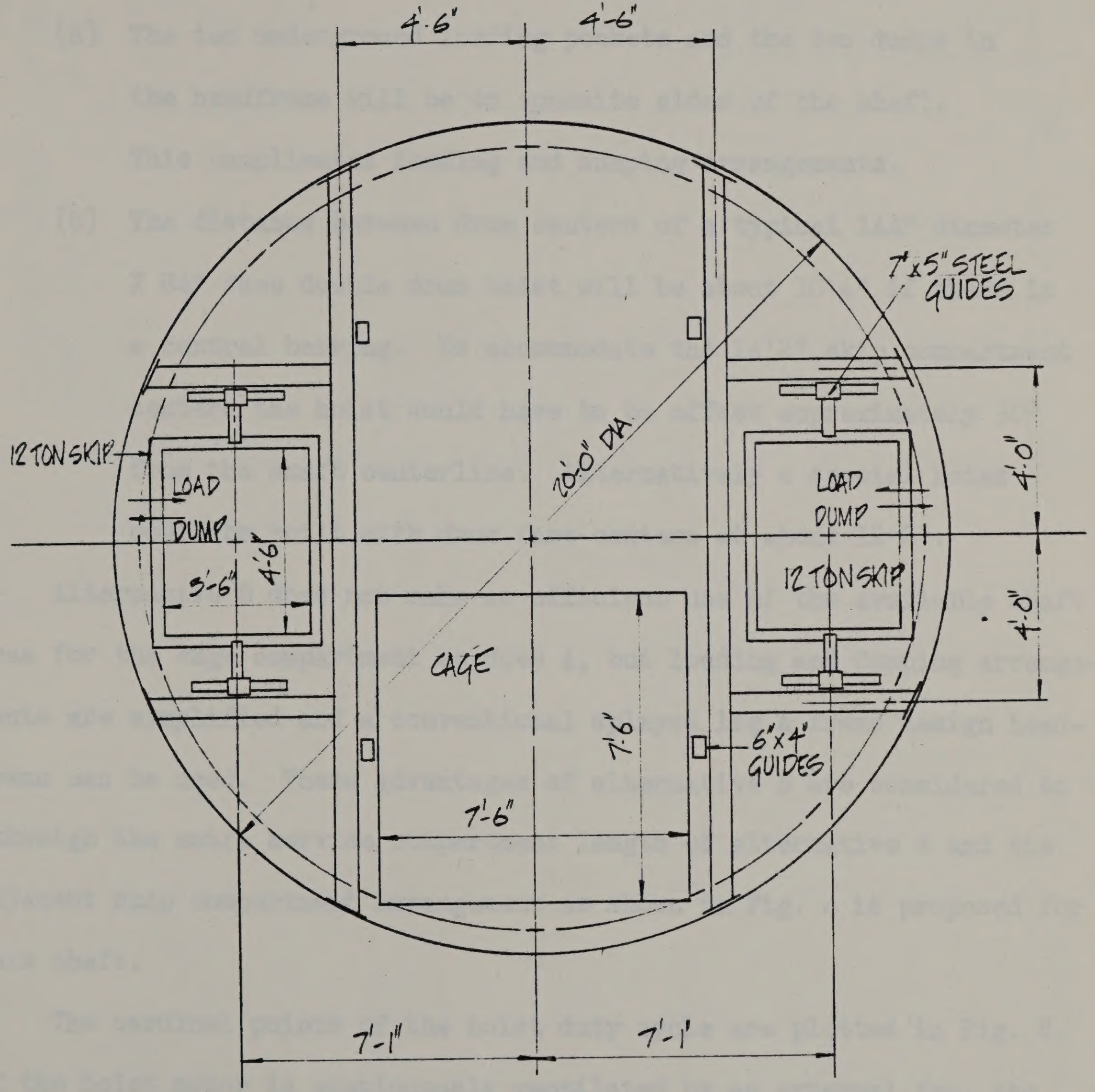
BASIC DIMENSIONS 12 TON SKIP
6'-6" BETWEEN GUIDES
SCALE 1/4" = 1 FT

TABLE 1

HOISTING DATA - DOUBLE DRUM PRODUCTION HOIST

Alternative	A	B
Hoisting Distance (feet)	 2500	. . . . .
Suspended Rope (feet)	 2550	. . . . .
HEADROPES		
Type	6 X 19 LANG LAY FLATTENED STRAND	
Diameter (inches)	. . . . 1.75	. . . . .
Ultimate Strength (lbs)	 303000	. . . . .
Weight/Foot (lbs)	. . . . 5.51	. . . . .
Drum/Rope Diameter	. . . . 82.1	. . . . .
SUSPENDED LOADS (lbs)		
Headrope	 14050	. . . . .
Payload	 24000	. . . . .
Skip Weight	 18000	. . . . .
ROPE TENSIONS (lbs)		
Full Skip at L. P.	 56050	. . . . .
Full Skip at Dump	 42000	. . . . .
ROPE FACTORS OF SAFETY		
Full Skip at L.P.	. . . . 5.41	. . . . .
Full Skip at Dump	. . . . 7.21	. . . . .
Statutory	. . . . 5.0	. . . . .
DRUM PARAMETERS		
Drum Diameter (inches)	144	144
Drum Face (inches)	84	60
Rope Pitch (inches)	1.84	1.84
Turns/Layer	45	32
Drum Revs/Cycle	64	64
Dead Turns	6	6
Total Turns	73	73
Layers of Rope	1.6	2.3
ROPE STRETCH WHEN LOADING ORE		
Load (P) (lbs)	 24000	. . . . .
Rope Length (L) (Feet)	 2550	. . . . .
Rope Area (A) (sq. ins.)	. . . . 2.40	. . . . .
Rope Modulus (E) (lbs/sq. in.)	. . . . 9 X 10^6	. . . . .
STRETCH = $\frac{P.L}{A.E}$ (feet)	. . . . 2.84	. . . . .

FIGURE 7



SHAFT LAYOUT ALTERNATIVE A

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compartment that is 7'8" between guides, but not symmetrical about the shaft centerline (Fig. 4).

Alternative A allows the optimum use of the shaft cross sectional area, but has the following disadvantages -

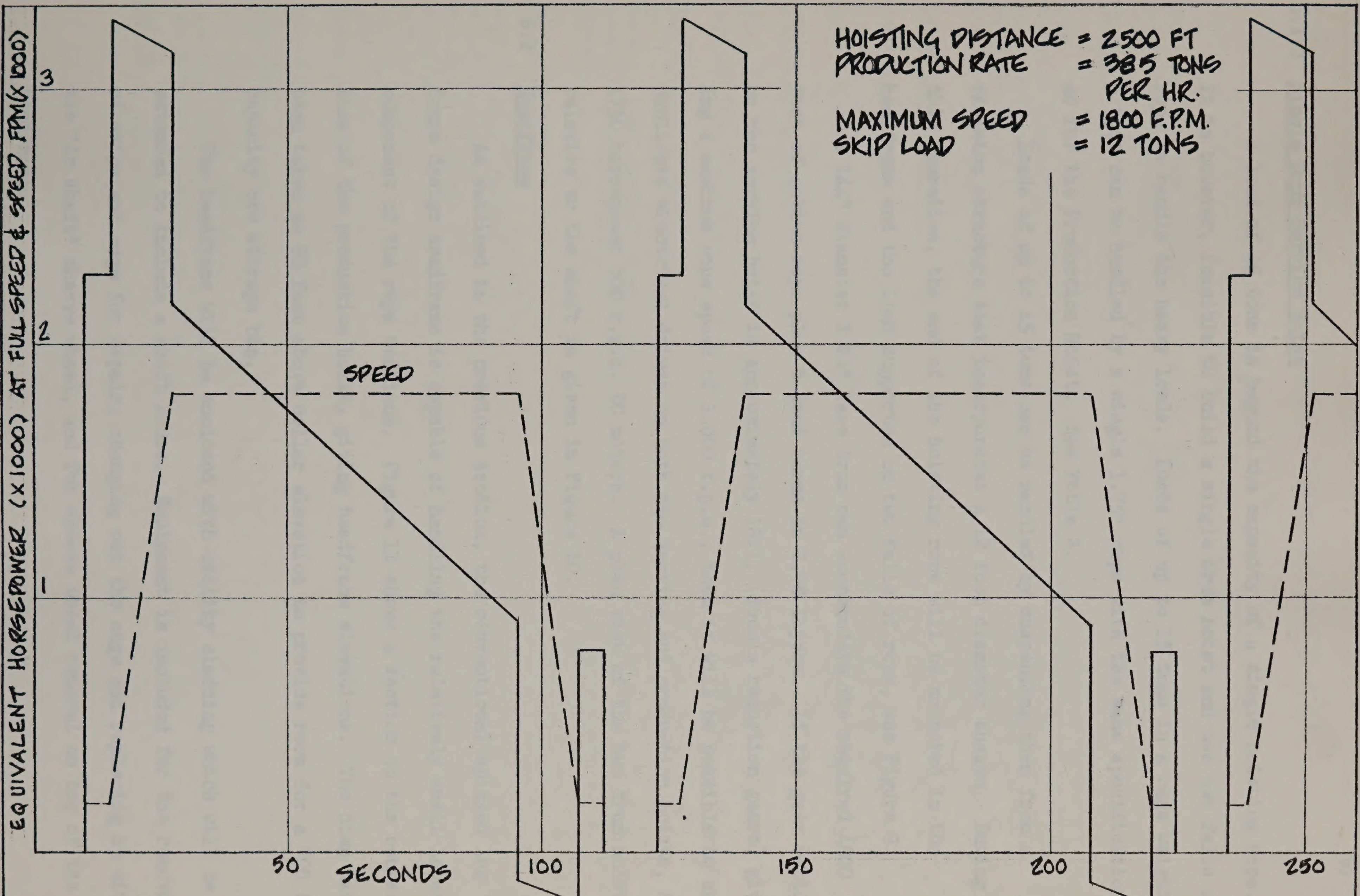
(a) The two underground loading pockets and the two dumps in the headframe will be on opposite sides of the shaft. This complicates loading and dumping arrangements.

(b) The distance between drum centers of a typical 144" diameter X 84" face double drum hoist will be about 10'4" if there is a central bearing. To accommodate the 14'2" skip compartment centers the hoist would have to be offset approximately 30° from the shaft centerline. Alternatively a special hoist could be built with drum face centers at about 12'2".

Alternative B does not make as efficient use of the available shaft area for the cage compartment as does A, but loading and dumping arrangements are simplified and a conventional splayed leg A frame design headframe can be used. These advantages of alternative B are considered to outweigh the extra service compartment length of alternative A and the adjacent skip compartment arrangement as shown in Fig. 4 is proposed for this shaft.

The cardinal points of the hoist duty cycle are plotted in Fig. 8. If the hoist motor is continuously ventilated by an external fan, the R.M.S. horsepower will be 1700. Peak horsepower during acceleration will be 3270 HP.

A single 1750 horsepower 500 r.p.m. motor capable of frequent 200% peaks during the acceleration period will be adequate for this duty. The gear reducer will be single reduction, approximately 10.5:1 ratio.



HOISTING DISTANCE = 2500 FT
PRODUCTION RATE = 385 TONS PER HR.
MAXIMUM SPEED = 1800 F.P.M.
SKIP LOAD = 12 TONS

DUTY CYCLE-DOUBLE DRUM HOIST / BALANCED SKIPS

FIGURE 8

6.1.3 Single Drum Service Hoist

A load of 45 tons is beyond the capacity of a single hoisting rope. It is however, feasible to build a single drum hoist and use two falls of rope to handle the heavy loads. Loads of up to 15 tons in a cage weighing 5 tons can be handled by a single 1.75" rope with the same specifications as for the Production Hoist. See Table 2.

Loads of up to 45 tons can be handled by suspending them from a skeleton structure that incorporates a 12 foot diameter sheave. During this operation, the end of the hoisting rope will be anchored in the headframe and the load supported on two falls of rope, see Figure 9.

A 144" diameter X 84" face drum can accommodate the required 4800 feet of active rope plus 6 dead turns in 2.98 layers. If the gear ratio on the service hoist is approximately 18:1, (double reduction gears) giving a maximum rope speed of 1,000 f.p.m., then it will be possible to use duplicate electrical drives on both the service and production hoists, i.e., 1750 horsepower 500 r.p.m. DC motors. A plan view of the two drum hoists relative to the shaft is given in Figure 10.

6.2 Headframe

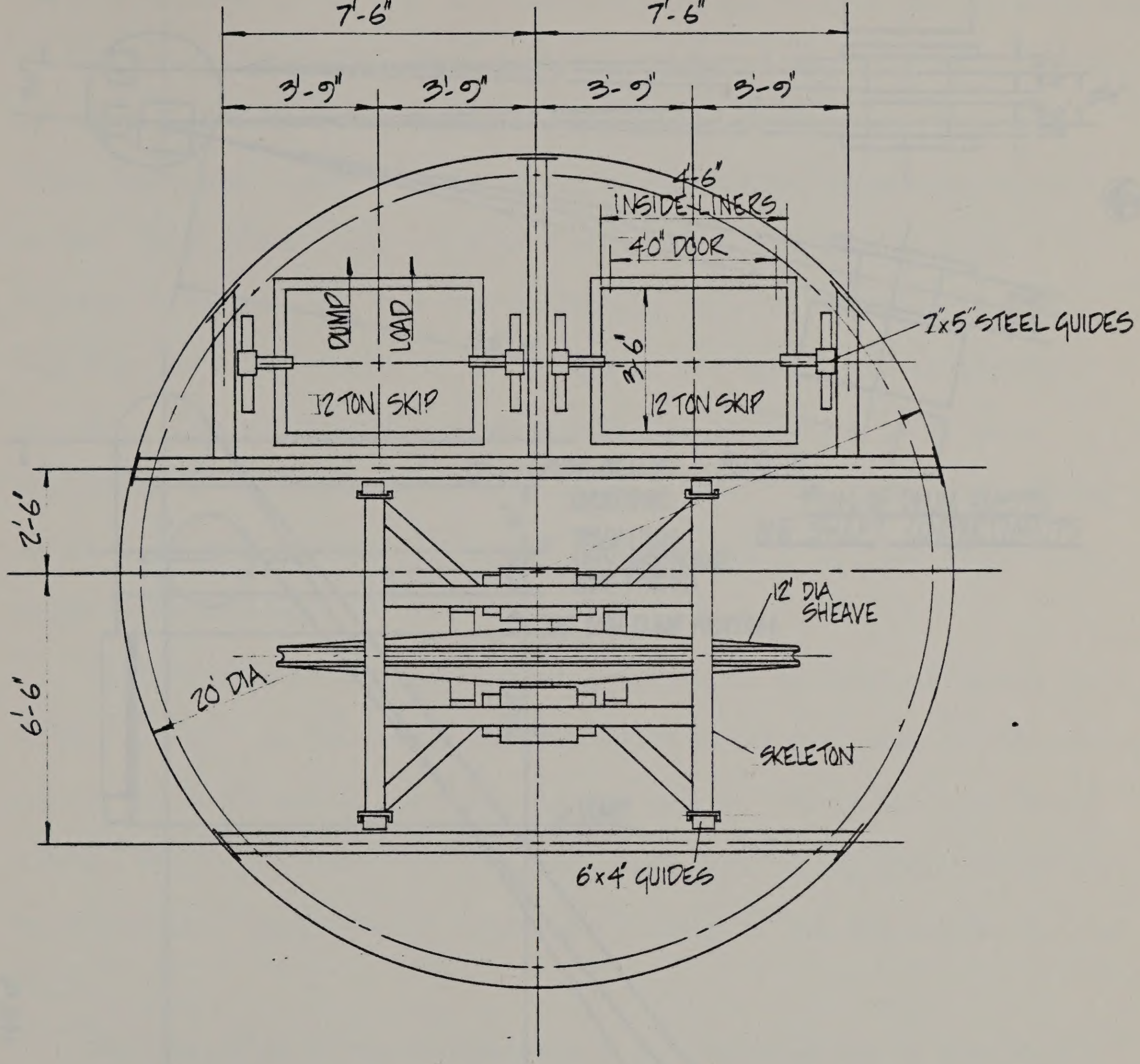
As outlined in the previous section, the conventional splayed leg 'A' frame design headframe is capable of handling the relatively small lateral component of the rope tensions. Figure 11 shows a section on the centerline of the production hoist, giving headframe elevations. The dump has been taken as 90 feet above collar elevation to provide room for a 700 ton capacity ore storage bin.

The headframe will be enclosed with utility cladding which will be extended to include a shaft house. Equipment is included for the removal of skips and cage for repair, changing out the cage and replacing it with the "in shaft" sheave wheel, and for sheave wheel removal on top of the headframe.

TABLE 2 HOISTING DATA - SINGLE DRUM SERVICE HOIST

	NORMAL LOAD	MAXIMUM MATERIAL (USING DOUBLE DOWN SHEAVE)
Hoisting Distance (feet)	 2500	. . . . . .
Suspended Rope (feet)	 2550	. . . . . .
HEADROPES		
Type	6 X 19 LANG LAY FLATTENED STRAND	
Diameter (inches)	1 X 1.75	2 X 1.75
Ultimate Strength (lbs)	1 X 303000	2 X 303000
Weight/foot (lbs)	1 X 5.51	2 X 5.51
Drum/Rope Diameter	82:1	82:1
SUSPENDED LOADS (lbs)		
Headrope	14050	2 X 14050
Material Load	30000	90000
Cage Weight	10000	15000
ROPE TENSIONS (lbs)		
Material Load at 2400 Level	54050	2 X 66550
Material Load at Collar	40000	2 X 52500
ROPE FACTORS OF SAFETY		
Material Load at 2400 Level	5.61	4.55
Material Load at Collar	7.57	5.77
Statutory	5.0	4.0
DRUM PARAMETERS		
Drum Diameter (inches)	 144	. . . . . .
Drum Face (inches)	 84	. . . . . .
Rope Pitch (inches)	. . . . 1.84	. . . . . .
Turns/Layer	 45	. . . . . .
Drum Revs/Cycle	64	128
Dead Turns	6	6
Total Turns	73	140
Layers of Rope	1.6	3.1
ROPE STRETCH WHEN LOADING MATERIAL		
Load (P) (lbs) X 10^3	30000	45000
Rope Length (L) (feet)	2550	2550
Rope Area (A) (sq. ins.)	2.4	2.4
Rope Modulus (E)(lb/sq. in)	9 X 10^6	9 X 10^6
STRETCH = $\frac{P.L.}{A.E}$ (feet)	3.54	5.31

FIGURE 9



SHAFT LAYOUT WHEN HANDLING HEAVY LOADS

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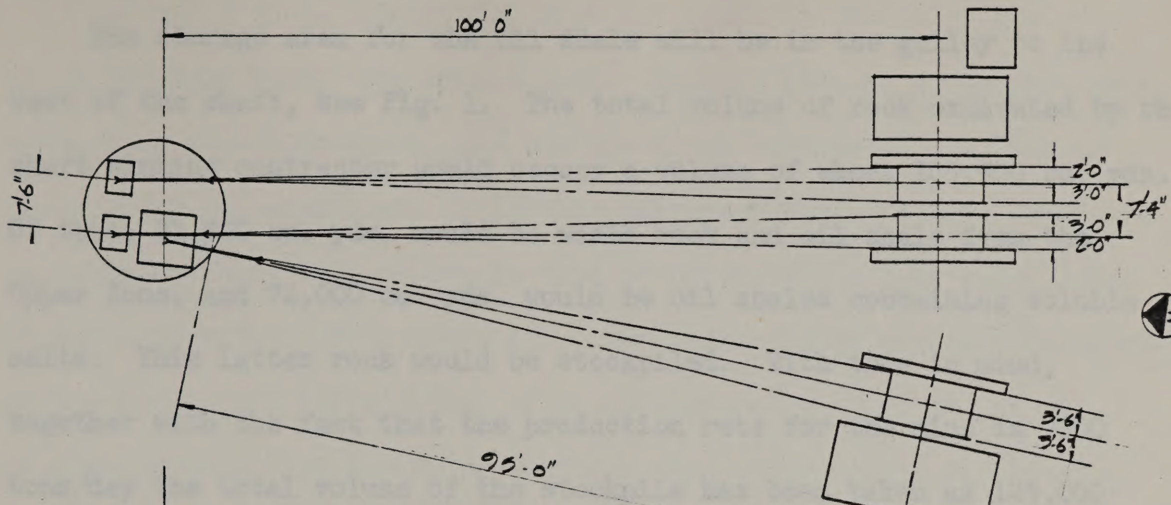
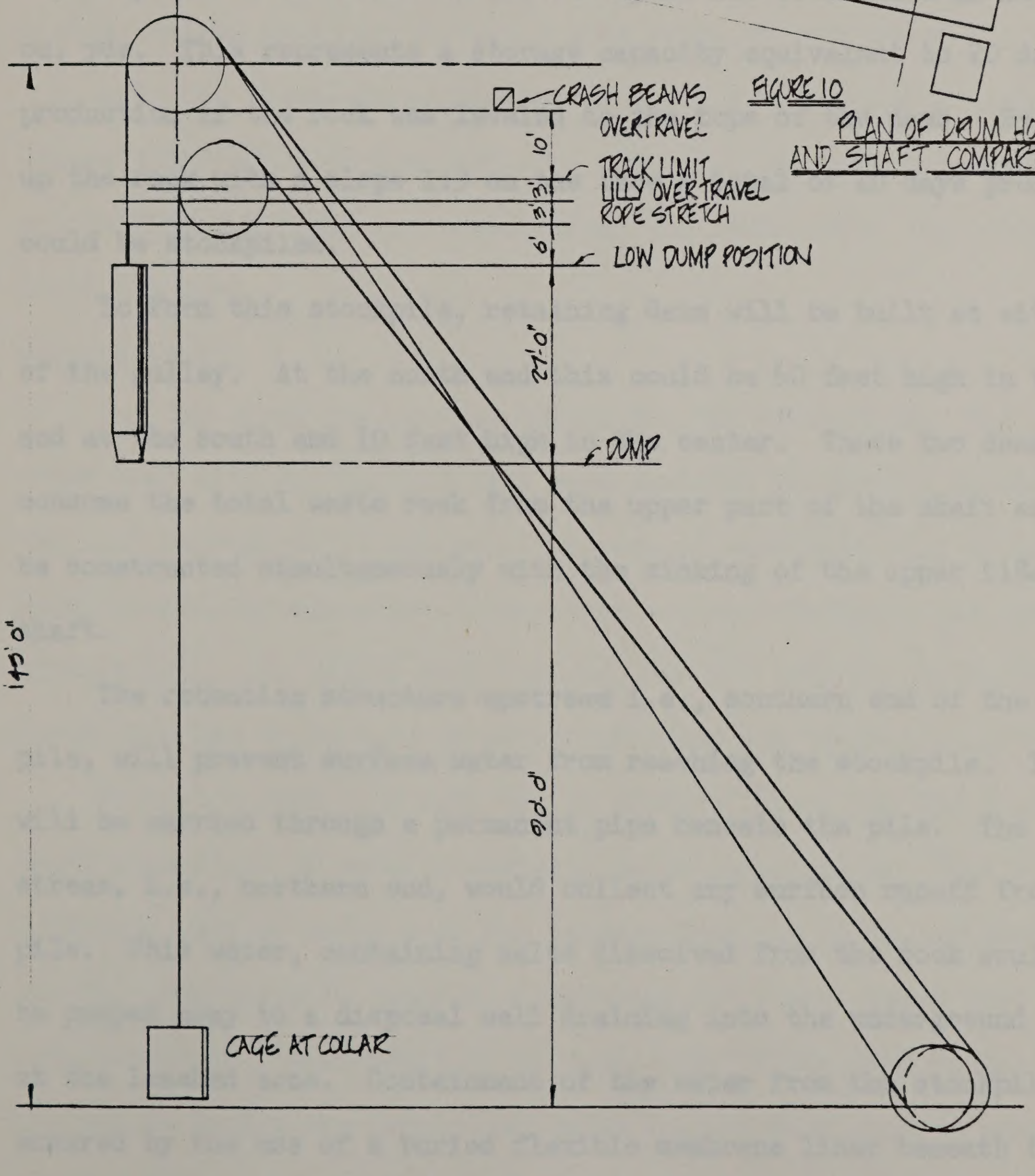


FIGURE 10
PLAN OF DRUM HOISTS
AND SHAFT COMPARTMENTS



ELEVATION FIGURE 11

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6.3 Surface Ore Removal and Storage

The storage area for the oil shale will be in the gulley to the west of the shaft, see Fig. 1. The total volume of rock excavated by the shaft sinking contractor would occupy a volume of about 105,500 cu. yds. Of this, 31,500 cu. yds. would be waste rock and oil shale from the Upper Zone, and 74,000 cu. yds. would be oil shales containing soluble salts. This latter rock would be stockpiled. With this in mind, together with the fact that the production rate for the mine is 7500 tons/day the total volume of the stockpile has been taken as 125,000 cu. yds. This represents a storage capacity equivalent to 20 days production if the rock was leveled to the tops of the dams. By heaping up the rock with a slope 1:3 on the face a total of 40 days production could be stockpiled.

To form this stockpile, retaining dams will be built at either end of the gulley. At the north end this could be 60 feet high in the center and at the south end 10 feet high in the center. These two dams would consume the total waste rock from the upper part of the shaft and would be constructed simultaneously with the sinking of the upper 1184 feet of shaft.

The retention structure upstream i.e., southern end of the stockpile, will prevent surface water from reaching the stockpile. This water will be carried through a permanent pipe beneath the pile. The dam downstream, i.e., northern end, would collect any surface runoff from the pile. This water, containing salts dissolved from the rock would initially be pumped away to a disposal well draining into the underground aquifer at the leached zone. Containment of the water from the stockpile would be ensured by the use of a buried flexible membrane liner beneath the pile.

The ore will be transported from the shaft to the stockpile area in off-highway heavy duty dump trucks. Amongst others, a conveyor belt system was also considered, but was discarded because of its lack of flexibility and high cost. A further advantage that the dump truck has over the conveyor belt system is that it can handle a much wider range of particle sizes. This could be an important factor during development work prior to the installation of an underground crusher or, if no crusher is installed during production, especially as the different mining methods test may produce a wide range of rock sizes.

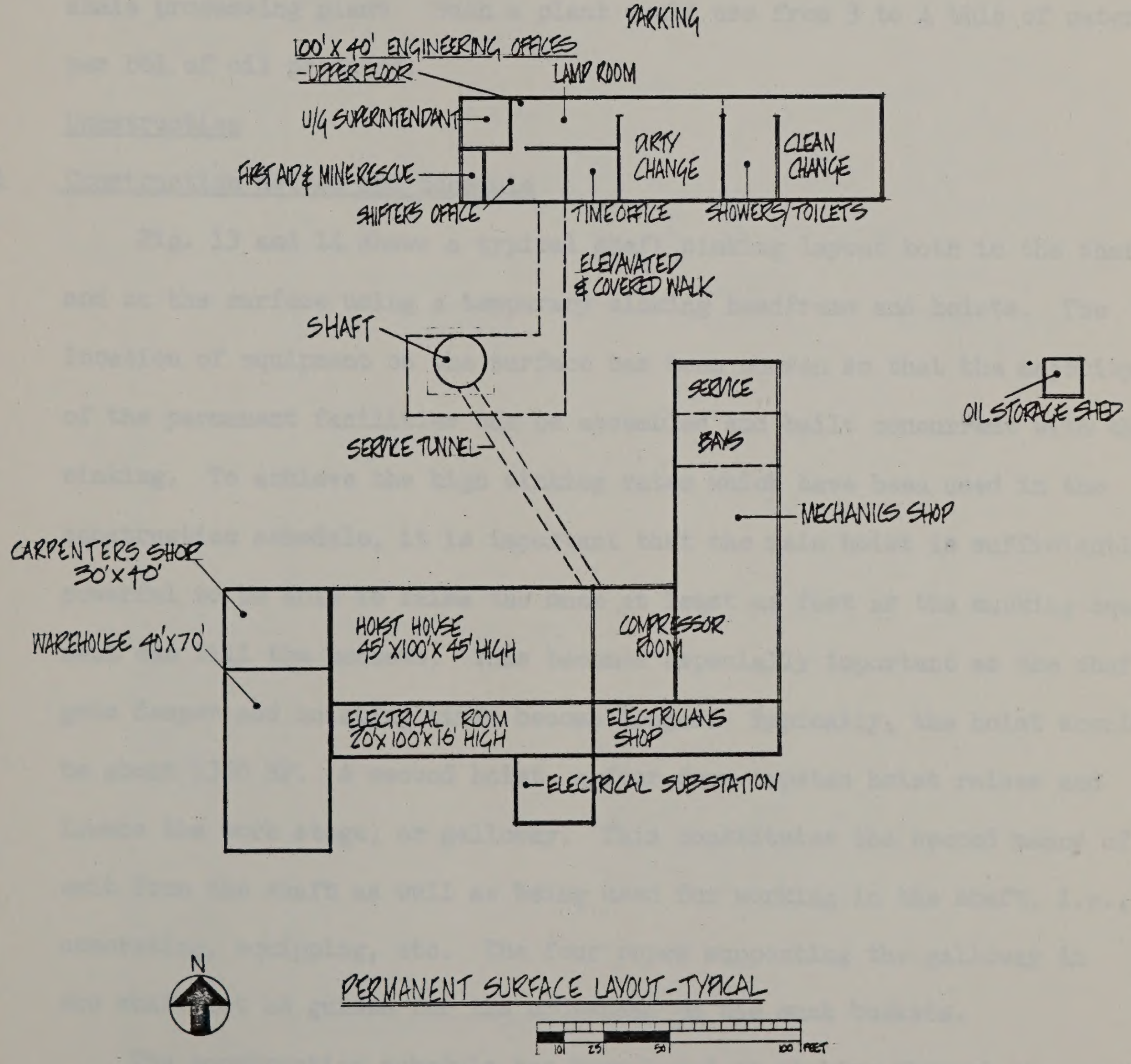
A 30 ton truck can handle the production requirements although a second truck would be required as a spare for use during downtime and maintenance. A bulldozer would spread the rock across the width of the stockpile area.

6.4 Surface Layout

Fig. 12 shows the surface layout for the Pilot Mine. The main facilities that will be required have been included in what is considered to be the most convenient locations. The size of each unit will depend upon the requirements for the mine. Items such as the hoisthouse and headframe and offices etc., are fairly well defined, but for others, reasonable assumptions have had to be made to arrive at sizes and therefore costs.

Provision should be made on the surface layout for the disposal of small quantities of water containing salts in solution. This requires an injection well or wells into the Leached Zone. A 12 inch diameter cell, 300 ft. long would be capable of handling 100 g.p.m. using just the static head available. This would be suitable most of the time during shaft sinking and to handle the precipitation on the ore storage area especially as

FIGURE 12



PERMANENT SURFACE LAYOUT-TYPICAL

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DRWN BY N. HYND

the settling pond would act as a surge tank. However, provision would be made for pumping the water into the well and in this way, the injection well could handle quantities up to 500 g.p.m. using excess pressures up to 200 psi.

If, at a later date, larger quantities of water were required to be pumped from the mine, it would be totally consumed, with no planned discharge to surface water bodies or to ground water aquifers in the oil shale processing plant. Such a plant would use from 3 to 4 bbls of water per bbl. of oil produced.

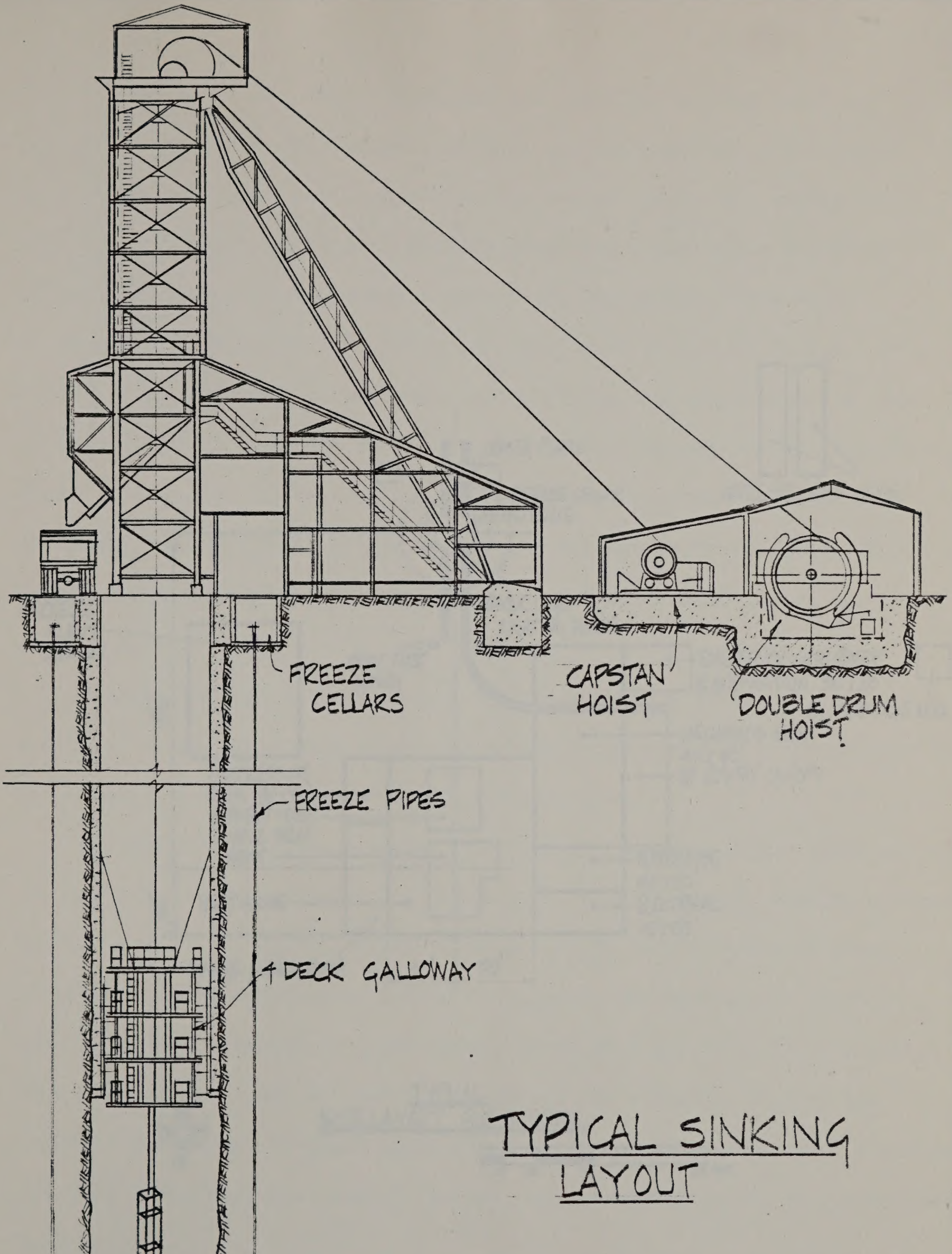
7. Construction

7.1 Construction Method and Schedule

Fig. 13 and 14 shows a typical shaft sinking layout both in the shaft and on the surface using a temporary sinking headframe and hoists. The location of equipment on the surface has been chosen so that the majority of the permanent facilities can be assembled and built concurrent with the sinking. To achieve the high sinking rates which have been used in the construction schedule, it is important that the main hoist is sufficiently powerful to be able to raise the muck at least as fast as the mucking equipment can fill the buckets. This becomes especially important as the shaft gets deeper and hoisting times become longer. Typically, the hoist should be about 1350 HP. A second hoist, a four drum capstan hoist raises and lowers the work stage, or galloway. This constitutes the second means of exit from the shaft as well as being used for working in the shaft, i.e., concreting, equipping, etc. The four ropes supporting the galloway in the shaft act as guides for the crosshead on the muck buckets.

The construction schedule has been based on sinking through the upper Evacuation Creek Member using a hydraulic breaker and mucking with a

FIGURE 13



TYPICAL SINKING LAYOUT

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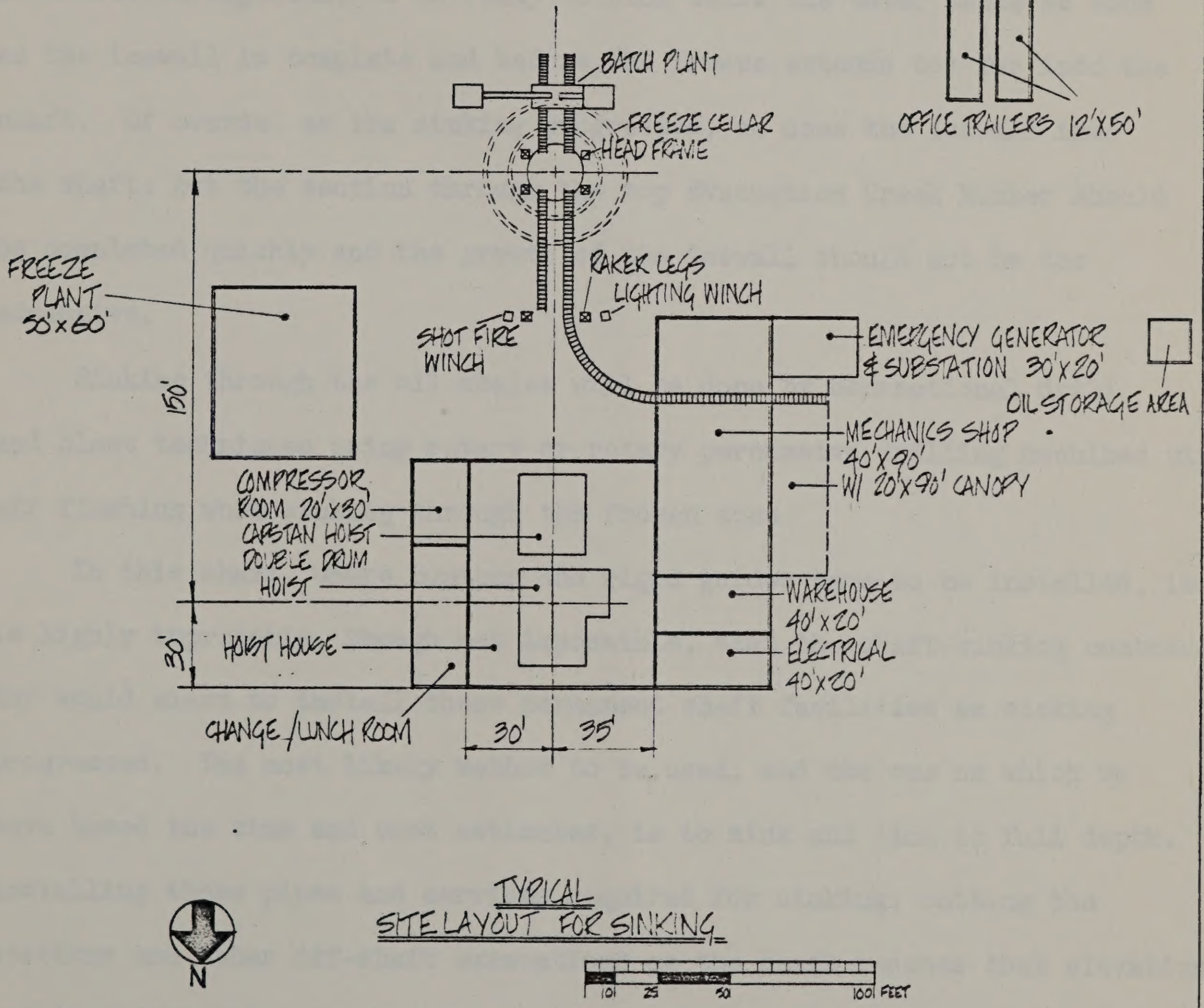
FIGURE 14



TYPICAL
SITE LAYOUT FOR SINKING

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DRWN BY N.HYND

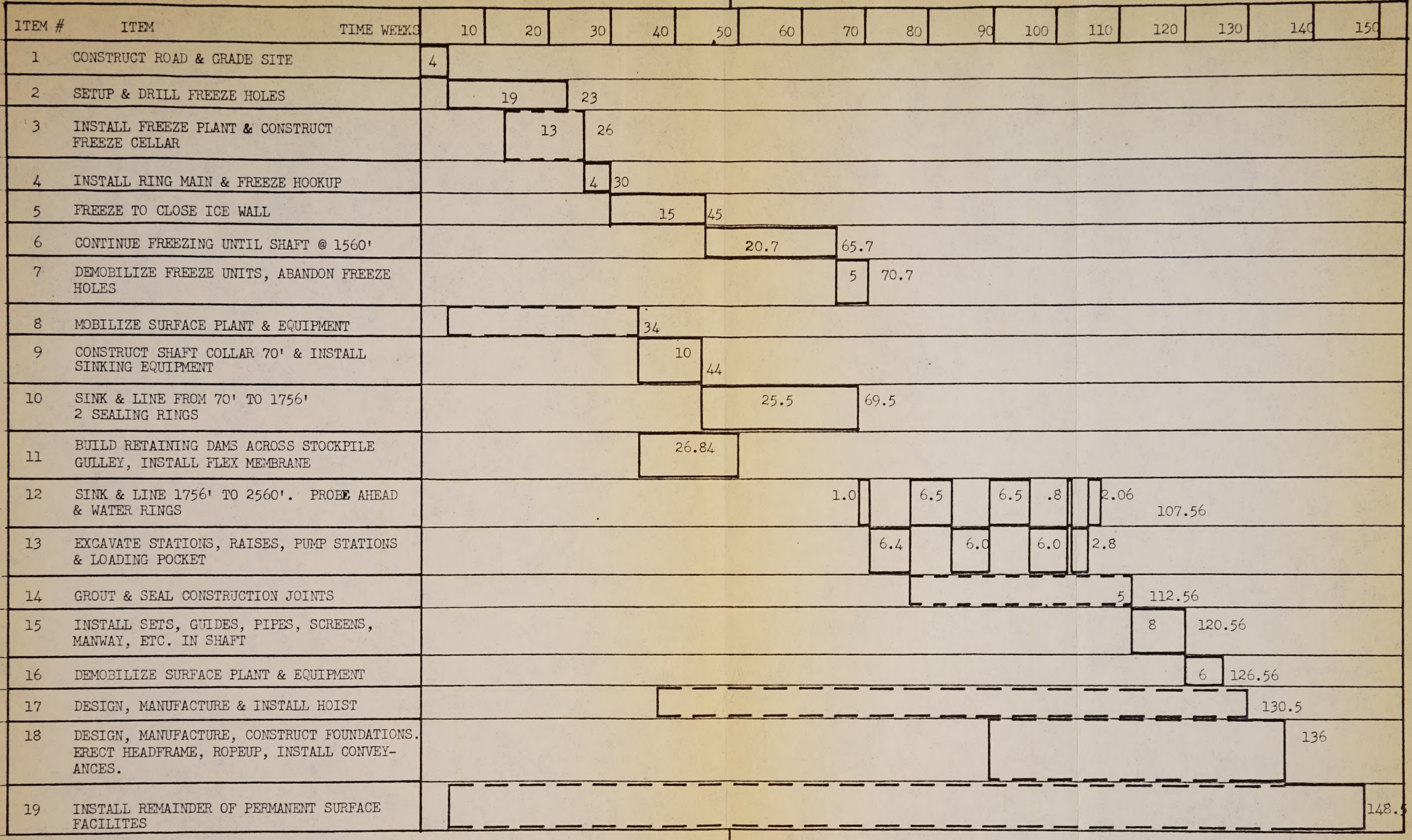
Cryderman Mucker. All the indications are that this system could be used to advantage in this silty sandstone and the time and costs could be reduced. However, particularly with a frozen shaft, the timing of each phase is very important. Sinking below the water table cannot be easily achieved until the icewall around the periphery of the shaft is closed, but the rock will break more easily if it is not frozen. It is therefore important to be ready to sink below the water table as soon as the icewall is complete and before the freeze extends too far into the shaft. Of course, as the sinking progresses, so does the icewall into the shaft, but the section through the top Evacuation Creek Member should be completed quickly and the growth of the icewall should not be too excessive.

Sinking through the oil shales will be done by conventional drill and blast techniques using rotary or rotary percussive drilling machines with air flushing when sinking through the frozen zone.

In this shaft, where buntons and rigid guides have to be installed, it is highly improbable, though not impossible, that the shaft sinking contractor would elect to install these permanent shaft facilities as sinking progressed. The most likely method to be used, and the one on which we have based the time and cost estimates, is to sink and line to full depth, installing those pipes and services required for sinking, cutting the stations and other off-shaft excavations as the shaft reaches that elevation, and then installing the buntons, guides, and remaining pipes in a separate operation.

Fig. 15 is the construction schedule for the sinking of the shaft and the erection of the permanent facilities.

CONSTRUCTION PROGRAM FIGURE 15



The initial phase is to establish access to the site and to level and grade the mine site. On the completion of this work, preparation can be made for drilling the freeze holes. During the drilling of these holes, the freeze plants are installed, and ready to be put into operation immediately the holes are equipped. A period of freezing is then required to close the icewall surrounding the shaft excavation. However, during this period, preparations for full scale sinking can be made by constructing the shaft collar to about 70 ft. and installing the sinking equipment. As indicated earlier, the timing here is important as the icewall should just be formed by the time the sinking reaches 120 ft. (the water table). Sinking, lining and freezing are then carried out until the shaft is below the fracture base in the Lower Zone, when freezing is discontinued. Sinking and lining is continued to the total depth.

Abandonment of the freeze holes and demobilization of the freeze plant can be started at the end of the freezing period with the work scheduled for completion before the start of the installation of the permanent facilities.

The back wall injection will be carried out and the lining construction joints sealed on the upper frozen part of the shaft when conditions allow, during the sinking of the bottom part of the shaft and the excavating of the stations.

The construction of the ore storage is phased in with the shaft sinking. The retention dams are formed with the rock from the upper part of the shaft, the flexible membrane is laid and buried, and the ore from the lower part of the shaft is put into the storage area thus formed.

After the shaft bottoms out, the permanent in-shaft facilities are installed. Maximum use should be made of the sinking contractors equipment for this work. This will include installing the furnishing steel,

power cables, service ranges, loading pocket equipment and installing the cage and skips in position in the shaft bottom in preparation for connection to the permanent hoist ropes. Consideration should also be given to the lowering of large bulky items of equipment down the shaft for use in the Pilot Mine prior to the equipping of the shaft.

On completion of this work, the sinking equipment will be demobilized and the other remaining permanent facilities are constructed and installed. During these final stages of the work specialized equipment and personnel will be used, i.e., for hoist erection and headframe construction. The major portion of this work would be carried out on a subcontract basis using subcontractors specializing in the type of work to be done.

The estimated construction time for the completed shaft is 126.5 weeks. This schedule is based on 3 shift working, 7 days per week by sinking crews only for the whole period except for mobilization and sinking the collar, and demobilization, which would be on a 1 shift basis. The completion of the remaining permanent facilities is estimated to take a further 22 weeks.

7.2 Manpower

The manpower requirement is given in Figure 16 and will vary throughout the project depending upon the stage of the project. However, it should be consistent during the sinking of the lower part of the shaft at 76 except for the addition of 2 raise miners during the station work. During this phase of the work, the size of the work force will be close to the maximum required at any time during the project. Appendix IV gives the employee classification and the wage rates used in estimating construction costs. It will not

FIGURE 16

EMPLOYEE CLASSIFICATION SHAFT SINKING

EMPLOYEE WEEKS	0 4	13	30	34	44	55.2	65.7	70.7	107.56	120.5	126.5
PROJECT MANAGER		1	1	1	1	1	1	1	1	1	1
PROJECT SUPERINTENDENT		1	1	1	1	1	1	1	1	1	1
MECHANICAL SUPERINTENDENT		1	1	1	1	1	1	1	1	1	1
SENIOR ENGINEER		1	1	1	1	1	1	1	1	1	1
WALKERS		4	4	4	4	4	4	4	4	4	4
ASSISTANT ENGINEER		1	1	1	1	1	1	1	1	1	1
LEAD MECHANIC		1	1	1	1	1	1	1	1	1	1
LEAD ELECTRICIAN		1	1	1	1	1	1	1	1	1	1
SURFACE FOREMAN		1	1	1	1	1	1	1	1	1	1
FREEZE PLANT SUPERINTENDENT			1	1	1	1	1	1			
ACCOUNTANT		1	1	1	1	1	1	1	1	1	1
SECRETARY		1	1	1	1	1	1	1	1	1	1
WAREHOUSEMAN		1	1	1	1	1	1	1	1	1	1
LEAD MINER					4	4	4	4	4	4	
MINERS					4	8	16	16	18	16	
MUCKING UNIT OPERATORS					2	8	8	8	8	8	
FREEZE PLANT OPERATOR			3	4	4	4	4	2			
DECKMEN					1	8	8	8	8	8	
HOISTMAN						4	4	4	4	4	1
SHIFT MECHANICS		4	4	4	4	4	4	4	4	4	4
SHIFT ELECTRICIANS		4	4	4	4	4	4	4	4	4	2
OTHER MECHANICS		2	2	2	2	2	2	2	2	2	2
WELDERS, DRILL DOCTORS			1	1	1	1	1	1	1	1	1
POWDERMAN				1	1	1	1	1	1	1	
DRY ATTENDENT/FIRST AID		1	1	1	1	1	1	1	1	1	1
CRANE OPERATOR		1	1	1	1						
CARPENTER		1	1	1	1	1	1	1	1		
FRONT END LOADER OPERATOR		1	1	1	1	1	1	1	1		
BATCH PLANT OPERATOR		1	1	1	1	1	1	1	1		
LABORERS				7	3	2	2	2	2	2	4

be possible to recruit labor locally in either the numbers or with the skills required and the major part of the force will need to be brought in from other areas. There is limited accommodation available within driving distance of the mine, and provision should be made to accommodate part of this non-local labor in a camp located at or near the site. In our cost estimate we have allowed that a camp which can accommodate up to 50% of the project labor force during sinking will be provided.

A breakdown of camp costs is given in Appendix IV.

7.3 Procurement of Equipment

7.3.1 Contractors

The two areas where a shaft sinking contractor could encounter problems is in procuring the freeze hole casings and the freeze plant. Both of these items are needed at the start of the work.

In 1974, freeze hole casing was in short supply, but the situation has eased since then. It is, however, recommended that immediately a decision has been made to proceed with the shaft sinking steps should be taken to secure a supply of suitable casing which can then be used by the shaft sinking contractors.

Freeze plants are on a two year delivery, and it has been assumed that the shaft sinking contractor will either own the units needed or have a supply source for used units.

7.3.2 Permanent Facilities

The various component parts making up the permanent facilities are interdependent and the complete system must be designed as a unit. From the point of view of time the design and supply of the production hoist is the critical element, and a period of 18 months to two years must be allowed for this section. The headframe and other facilities

such as skips, cages, ropes, etc. would be designed in conjunction with the hoist, although their procurement would take less time. As for skips, cages, ropes and loading and measuring pockets, orders should be placed a minimum of one year before they are required. For this reason it was considered more practical to use a sinking headframe and hoist for shaft sinking rather than the permanent headframe and hoist and the cost estimate is based on this condition. The headframe delivery time is 9 months. Neither of these items are required to be installed and operational until after the sinking is completed. The construction schedule indicates when these items should be ordered.

It should be stressed that, the above delivery times are based on 1st quarter 1976 supply conditions. They are by no means stable and as the project progresses the procurement position should be reviewed and updated.

8. Costs

8.1 Breakdown of Construction Costs

The cost for sinking the shaft alone would be $17,439,000. The total cost for the shaft together with the surface permanent facilities would be $23,422,000. These costs have been escalated allowing for a construction starting date of July 1978. The "Cost Estimate Breakdown" is presented by Figure 17 (Sheets 1 and 2).

8.2 Explanatory Notes on Cost Breakdown

(a) Labor

The cost of contractor's labor over the period of the contract. The labor rates used in the estimate are those for the year July 1978 to July 1979, as defined in Cementation's agreement with the International Union of Operating Engineers and Laborers International.

FIGURE 17 Sheet 1 of 2

U.S.B.M. PICEANCE CREEK SHAFT STUDY COST ESTIMATE BREAKDOWN

DESCRIPTION	Estimated Quantity	Unit	Labor	Materials	Consumable Materials	Internal Equipment Rental	External Equipment Rental	Subcontract	Power	Camp	Total Dollars	Unit
1 SITE PREPARATION												
(a) CONSTRUCT ROADS	1200	Yds.						26,400			26,400	22.00
(b) LEVEL GRADE & DRAIN SITE	12000	Cu.Yds.						39,600			39,600	3.30
2 FREEZE HOLE DRILLING												
(a) CONSTRUCT DRILLING PAD	L.S.							33,000			33,000	
(b) SET UP & DRILL FREEZE HOLES & OBSERVATION HOLES	46400	Ft.						2,118,600			2,118,600	45.65
3 FREEZING PREPARATIONS												
(a) INSTALL FREEZE PLANT, RING MAIN & FREEZE HOOKUP	L.S.		220,800	73,000	248,500		21,100		770	26,800	590,970	
(b) CONSTRUCT FREEZE CELLAR	L.S.							33,000			33,000	
4 FREEZING OPERATIONS	L.S.		268,600		65,100	162,000		132,000	132,500	25,000	785,200	
5 COMPLETION OF FREEZING												
(a) DEMOBILIZE FREEZE UNITS	L.S.		29,200		7,100		21,100		300	2,500	60,200	
(b) ABANDON FREEZE HOLES	L.S.							319,000			319,000	
6 MOBILIZE SURFACE PLANT & EQUIPMENT												
(a) SURFACE UTILITIES (PERMANENT)	L.S.							154,200			154,200	
(b) INSTALL SINKING EQUIPMENT	L.S.		653,900		175,800		75,600		5,200	90,200	1,000,700	
7 CONSTRUCT SHAFT COLLAR 0' TO 70'	70	Ft.	389,400	41,400	103,000	16,200			2,100	38,600	590,700	
8 (a) SINK 70' to 1756'	1686	Ft.	1,061,300		397,100	96,800			77,800	92,000	1,725,000	1,023.13
(b) LINE 70' to 1756'	1686	Ft.	454,800	1,108,500	161,000	41,500			33,300	39,400	1,838,500	1,090.45
9 INSTALL SEALING RINGS	2	NO	64,600	13,100	15,600	5,700			4,500	5,500	109,000	54,500.00
10 ORE STORAGE FACILITY (SURFACE)												
(a) BUILD RETAINING DAMS	INCLUDED IN ITEM 8a											
(b) INSTALL FLEXIBLE MEMBRANE & NECESSARY DRAINAGE	180000	Sq.Ft.						85,000			85,000	.47
11 (a) SINK 1756' to 2560'	804	Ft.	759,600		248,000	63,600			51,100	65,600	1,187,900	1,477.48
(b) LINE 1756' to 2560'	804	Ft.	325,600	395,200	103,600	27,300			21,900	28,100	901,700	1,121.51
(c) PROBE AHEAD	910	Ft.	50,700		12,300	4,300			3,400	4,400	75,100	82.52
(d) INSTALL WATER RINGS	4	NO	18,300	17,000	4,400	1,200			900	1,200	43,000	10,750.00
12 DEVELOPMENT WORK												
(a) STATION EXCAVATION	472400	Cu.Ft.	945,400		349,500	93,400			61,500	81,600	1,531,400	3.24
(b) STATION STEEL	75	Tons	96,100	119,900	23,300	9,600			6,200	8,300	263,400	3,508.00
(c) STATION CONCRETE	420	Cu.Yds	120,200	30,500	35,200	12,000			7,800	10,400	216,100	514.52
(d) INSTALL PUMPS SWITCHGEAR, ETC.	L.S.		66,800	205,900	16,200	6,700			4,300	5,800	305,700	
13 LOADING POCKET												
(a) EXCAVATION	10000	Cu. Ft.	32,000		10,300	3,200			2,000	2,800	50,300	5.03
(b) STRUCTURAL STEEL	25	Tons	32,000	40,000	7,800	3,200			2,000	3,600	88,600	3,544.00
(c) MEASURING POCKETS, ETC.	50	Tons	47,400	145,300	11,500	4,800			3,100	2,800	214,900	4,298.00
(d) CONCRETE	100	Cu. Yds	42,000	7,300	13,200	4,200			2,700	4,100	73,500	735.00
(e) BRING INTO OPERATION	L.S.		33,400		8,100	3,400			2,200	2,900	50,000	

FIGURE 17 SHEET 2 of 2

U. S. B. M. PICEANCE CREEK SHAFT STUDY COST ESTIMATE BREAKDOWN CONT.

DESCRIPTION	Estimated Quantity	Unit	Labor	Materials	Consumable Materials	Internal Equipment Rental	External Equipment Rental	Subcontract	Power	Camp	Total Dollars	Unit
14 RAISES												
(a) 6' X 6'	405	Ft.	21,700		10,500	4,800			1,400	1,900	40,300	99.50
(b) 8' X 8'	285	Ft.	27,200		13,100	6,000			1,800	2,300	50,400	176.84
(c) ORE POCKET U/G	25,080	Cu.Ft.	37,400		18,100	8,200			2,400	3,200	69,300	2.36
15 SEAL CONSTRUCTION JOINTS IN SHAFT LINING	75	NO.	322,500		78,100	28,300			36,300	28,500	493,700	6582.66
16 SHAFT EQUIPPING												
(a) SHAFT STEEL & GUIDES	549.525	Tons	516,000	877,930	125,000	45,200			27,200	45,700	1,637,030	2978.99
(b) SHAFT PIPE				369,300							369,300	
17 DEMOBILIZE SINKING EQUIPMENT	L.S.		165,900		40,200		39,600		1,100	21,900	268,700	
18 DESIGN, MANUFACTURE & INSTALL HOISTS, ROPES & CONVEYANCES												
(a) DOUBLE DRUM PRODUCTION HOIST	L.S.							1,777,000			1,777,000	
(b) SINGLE DRUM SERVICE HOIST	L.S.							1,484,100			1,484,100	
19 DESIGN MANUFACTURE & ERECT HEADFRAME INCLUDING ORE BINS, CLADDING, SERVICE CRANES, ETC.	L.S.							1,059,600			1,059,600	
20 PERMANENT FACILITIES												
(a) HOISTHOUSE W/FOUNDATIONS	L.S.							525,400			525,400	
(b) OFFICES/CHANGE ROOM	L.S.							151,000			151,000	
(c) SHOPS/WAREHOUSE	L.S.							135,200			135,200	
(d) ELECTRICAL SUBSTATION (SURFACE)	L.S.							237,600			237,600	
(e) OTHER SURFACE PLANT	L.S.							237,600			237,600	
(f) ORE REMOVAL SYSTEM	L.S.							330,000			330,000	
(g) MISCELLANEOUS BUILDINGS/CONCRETE AREAS	L.S.							45,100			45,100	
TOTALS			$6,802,800	$3,444,330	$2,301,600	$ 651,600	$ 157,400	$8,923,400	$ 495,770	$645,100	$23,422,000	

(b) Materials and Consumable Materials

The cost of items such as general consumable stores, drill steels and bits, explosives and detonators, etc. Also included is the cost of contractor's equipment which is purposely made for, and completely written off against the project, together with shaft furnishing steel, pipes, cables, and concrete.

(c) Internal Equipment Rental

The rental charges for the contractor's construction plant.

(d) External Equipment Rental

This item covers the rental of equipment not owned by the shaft sinking contractor.

(e) Subcontract

The cost of work subcontracted by the main contractor to specialist contractors. This will include, for example, freeze hole drilling, instrumentation, supply and installation of permanent facilities (hoists, headframes, etc.)

All the costs involved in this section of the work, that is, labor, materials, plant rentals, etc., are included in the allowance.

(f) Power

The estimate of power costs is based on the H.P. utilization on the site at a cost of $0.018 per kw hour with a surcharge of 25% for demand.

(g) Camp

The cost of purchasing, setting up and running the camp facilities.

3. Starting Date

The costs are based on a July 1978 starting date.

4. Escalation

A cost escalation rate of 10% per annum has been assumed. The costs given have been escalated.

5. Contractor's Profit

The costs include an allowance for the contractor's profit.

6. Contingency

In shaft construction, contingency allowances are best made on the basis of lost construction time. Thus those costs which are related to time contain a contingency allowance, i.e., labor, plant rentals, etc.

A contingency provision is included in these costs.

APPENDIX I

Information supplied to Cementation Company by USBM.

Borehole logs by Birdwell as follows:

Corehole U.S.B.M. 01-A.

(a) Gamma - Ray Density

(b) Gamma - Ray Neutron

(c) 3 Dimensional Velocity Log

(d) Caliper Log

(e) Temperature Log

(f) Electric Log

Elastic Property Logs by Birdwell of coreholes U.S.B.M. 01, 01-A and 02-A. Core Index Records sheets for coreholes U.S.B.M. 01-A and U.S.B.M. 02-A. Descriptive Log of corehole U.S.B.M. 01-A giving formation elevations, fractured, brecciated and incompetent zones (from core description) and water data from drilling operations.

Preliminary results of Aquifer Tests Conducted.

Physical properties, U.S.B.M./AEC Colorado, Corehole No. 3 (Bronco BR-1).

Large scale (1:24000) map of relevant area of Rio Blanco County, Colorado showing extent of leases and private land.

Aerial photograph of same area.

Chemical analysis of ground water from coreholes U.S.B.M. 01, U.S.B.M. 01-A, and U.S.B.M. 02-A.

U.S.B.M. Shaft Feasibility Study

Assumed Strata Conditions at Shaft Site - Appendix II

The strata conditions are based on the report by Cementation Research Ltd. (Appendix III) and the logs from coreholes U.S.B.M. 01-A and U.S.B.M. 02-A. Depths are relative to proposed shaft collar elevation 6340 feet.

Alluvium

Depth - from 0 ft. to 20 ft.

Length - 20 ft.

Rock Type - Silts and clays

Structural Characteristics - was observed to stand in open cut to at least 15 feet

Hydrology - dry

Remarks - the thickness of alluvium could vary from 0 ft. to 140 ft. within relatively short distances, depending on whether the location is on a crest or in a valley.

Evacuation Creek Member - Uinta Formation

Depth - from 20 ft. to 822 ft.

Length - 802 ft.

Rock Type - fine grained silty sandstones

Structural Characteristics - contains fragment fractures and could present support problems during shaft sinking.

Hydrology - rock itself is not porous. Fractures will be water producing over the length of the member below the water table. Major flows likely to be encountered between 600 to 1000 ft. (i.e., into the top of the Upper Zone).

Remarks - The hydrological information on this section is not sufficiently detailed to make forecasts of likely water flows from specific areas within this aquifer. However, the information indicates that the flow into the shaft from this formation together with the top of the Upper Zone could be up to 1400 g.p.m.

Parachute Creek Member

Upper Zone - not including the Mahogany Zone.

Depth	- 822 ft. to 999 ft.
Length	- 177 ft.
Rock Type	- Marlstone with some interbedded silstone
Structural Characteristics	- Contains fractures but these are not of such a nature as to present support problems during shaft sinking.
Hydrology	- rock is not porous. Fractures will be water producing over the length of the member.
Remarks	- See remarks on Evacuation Creek Member.

Mahogany Zone

Depth	- from 999 ft. to 1184 ft.
Length	- 185 ft.
Rock Type	- Marlstone
Structural Characteristics	- Competent. No support problems during shaft sinking.
Hydrology	- Dry
Remarks	- None.

Leached Zone

Depth	- from 1184 ft. to 1504 ft.
Length	- 320 ft.
Rock Stype	- Marlstone
Structural Characteristics	- Fractured throughout. In this zone nahcolite has been leached by ground water and in some areas this has resulted in beds collapsing to give zones of 'rubble'. - Special steps will need to be taken during shaft sinking to support the excavation.
Hydrology	- Highly water producing throughout its length from fractures, leached and collapsed zones.
Remarks	- Detailed hydrological data is not available.

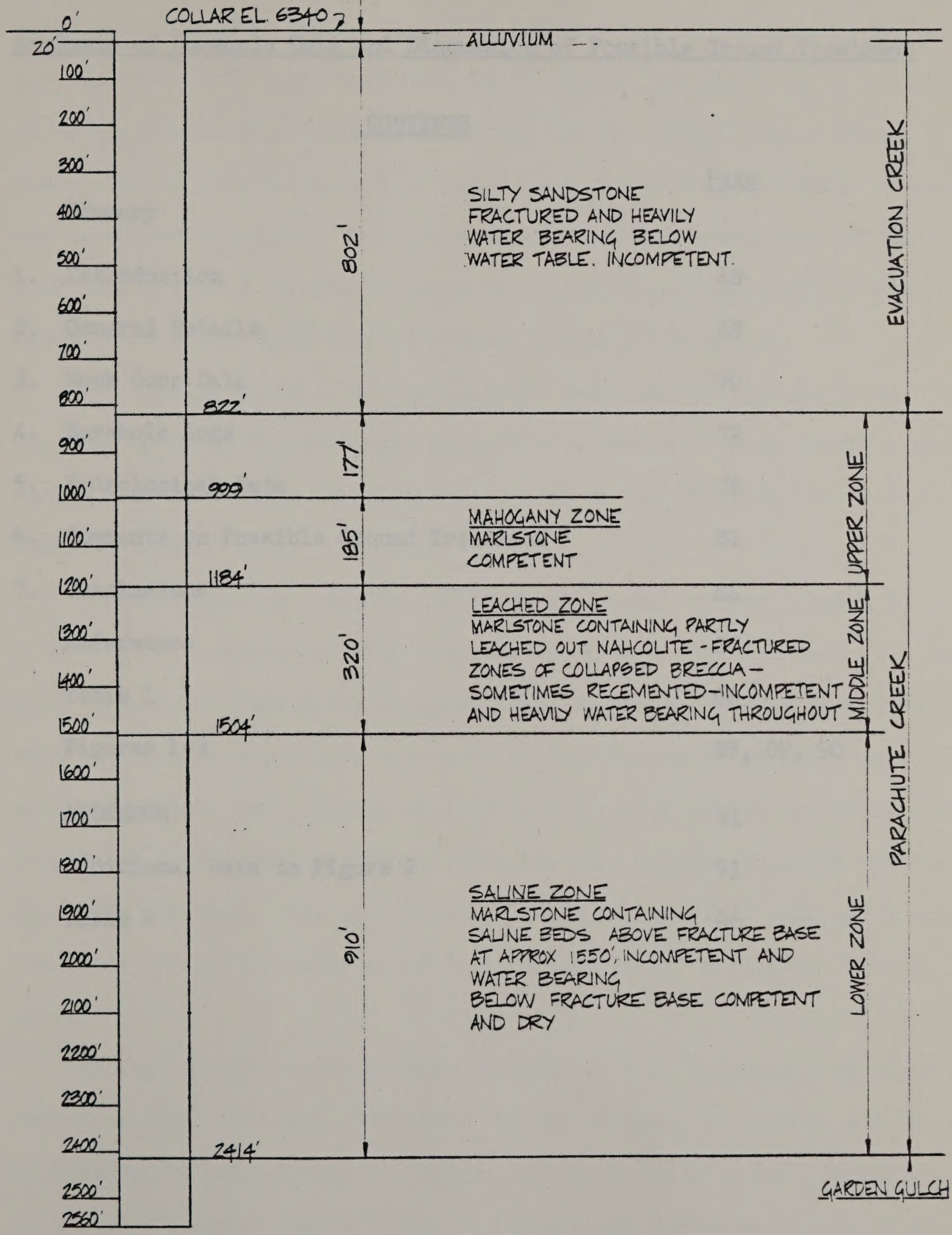
Leached Zone

Remarks (Continued)	- However, the available information indicates that the total flow into the shaft could be up to 5,000 gpm.

Lower Zone

Depth	- from 1504 ft. to 2414 ft.
Length	- 910 ft.
Rock Type	- Marlstone with zones of nahcolite. The mining levels are located in this zone.
Structural Characteristics	- The base of the fracturing could extend up to 150 ft. into the zone, however, there was little evidence in core 01-A of extensive fracturing into this zone. It is assumed to be 50 ft. The zone to the base of the fracturing will require special support consideration during shaft sinking. - The section below the base of the fracturing is competent and will not present a problem during sinking.
Hydrology	- The 50 ft. section above the base of the fracturing will be highly water producing from fractures. The remainder will be dry.
Remarks	- The data from C.H. 29 may allow a closer definition of the water producing properties of this section.

Attached is a section of the strata.



ASSUMED SECTION OF STRATA ON SHAFT CENTERLINE
SHOWING MAIN ROCK BEDS IN GREEN RIVER FORMATION

THE CEMENTATION COMPANY OF AMERICA INCORPORATED

DRWN BY NH

APPENDIX III

Analysis of Corehole Data and Discussion of Possible Ground Treatment

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Summary

This report is a detailed analysis of rock core, borehole log and water test data from two boreholes in the Piceance Creek Basin in Northwest Colorado. It forms part of a shaft feasibility study being conducted by the Cementation Company of America for the United States Bureau of Mines, and in turn forms part of their (i.e. U.S.B.M.), oil shale research and development program. The proposed shaft (20 feet diameter and about 2400 feet deep) is for the recovery of oil shale and associated sodium minerals from the thick Parachute Creek oil shale formation.

The information used in our analysis comprises preliminary data supplied by U.S.B.M., and supplementary data obtained by us during a core examination in Laramie, Wyoming and a visit to the Piceance Creek site in January, 1976.

To reach the ore deposits the shaft will have to pass through two major aquifers before entering competent rock at a depth of some 1500 feet. Apart from a 100 feet thick 'barrier' zone at about 950-1050 feet depth, the entire formation above 1500 feet must be regarded as potentially wet. We estimate a total water inflow to a 28 feet diameter excavation of between 10,000 and 20,000 gallons/minute, with the main inflow zones 500-900 feet and 1050-1400 feet. The rock in these zones is generally weak and highly fractured and in the lower aquifer contains frequent collapsed "rubble" zones.

To sink a 'dry' shaft ground treatment will be required from about 200 feet to a depth of about 1500 feet. On the evidence available, and subject to a number of conditions, we believe that freezing is a more feasible means of ground treatment than grouting in the present instance. Grouting would necessarily be very extensive and somewhat unpredictable especially since

ground strengthening as well as water stopping will probably be required in certain zones. A far more detailed "picture" of the formation is needed to design such an exercise. In-situ freezing and/or grouting trials are recommended prior to shaft sinking.

1. Introduction

As part of its oil shale research and development program the United States Bureau of Mines have awarded a prefeasibility study contract to the Cementation Company of America, Inc. This contract calls for preliminary layout, scheduling and costing for the construction of a single shaft, surface plant facilities, oil shale handling facilities and disposal, at a site in the Piceance Creek Basin, Colorado. The purpose of the study is to develop order of magnitude estimates of construction costs and operating funds required, and to determine the probable economic feasibility of the oil shale recovery project.

In order to select the most appropriate ground treatment method necessary to sink a 'dry' shaft, Cementation Research Ltd. were engaged by the Cementation Company of America to assist in the evaluation of rock and hydrological conditions. Some data from corehole 01-A was made available by U.S.B.M. in late December, 1975. This was supplemented by further information gained by Cementation personnel (S. H. E. Phillips and Dr. G. P. Daw) during an inspection of the 01-A core in Laramie and a visit to the Piceance Creek site, in mid January. During the site visit we witnessed coring of the latter stages of a second corehole 02-A, adjacent to 01-A and also in the vicinity of the proposed shaft.

The purpose of this report is to present and correlate all of the collected rock and hydrological information relevant to shaft sinking, and to comment on possible ground treatment methods.

2. General Details

The Piceance Creek Basin is situated in Rio Blanco County in Northwestern Colorado, lying between the White River (which it serves) to the North, and the Colorado River to the South. The proposed shaft site is in Horse Draw Creek (Fig. 1) which is some 12 miles South of the White River.

The high grade oil shale deposits occur in the Parachute Creek member of the Green River Formation which also extends into north-east Utah and southern Wyoming. However, an estimated 75% of the total reserves lie in the Piceance Creek Basin. They are unique in that they contain the rare minerals nahcolite ($NaHCO_3$) and dawsonite ($NaAl(OH)_2CO_3$) in amounts which are of potential economic value.

In the Horse Draw Creek vicinity, the Parachute Creek Oil Shales extend from a depth of some 800 ft. to 2400 ft. They are overlain by about 700 ft. of sandy siltstones of the Uinta Formation (Evacuation Creek Member) and underlain by illite shales of the Garden Gulch member. The oil shales which are basically well cemented, low permeability, dolomitic marlstones are divided into three main zones. The lower (or saline) zone is composed of competent oil shale, rich in kerogen and containing the main concentrations of nahcolite. The middle (or leached) zone consists of fractured oil shale with high porosity. Originally comprising periodically altering kerogen-rich to kerogen-lean marlstone with substantial amounts of salt it was subsequently fractured during periods of tectonic activity, particularly in regions of low oil content. Ground water found its way into the fractures and leached out the salts leaving a fractured porous oil shale, certain zones of which collapsed into rubble. It thus now provides a major aquifer for vast quantities of saline water. The Upper zone, some 300 to 600 feet thick is of higher grade oil shale and includes the particularly rich Mahogany zone in its lower half. The relatively high kerogen content of the Mahogany and Lower Zones is thought to be the reason that the fracturing and subsequent leaching did not extend far beyond the middle zone.

The geological divisions of the strata, as found in corehole 01-A, are indicated in the section contained in the overall log of Figure 2.

3. Rock Core Data

Corehole 01-A was drilled and cased to a diameter of 13-3/8 inches through some 140 feet of alluvium to a depth of 180 feet. Below the casing a 9-7/8 inch drill was used through the Uinta siltstones and into the Upper Oil Shales to a depth of 830 feet. Drill cuttings were retrieved and analyzed from this section. A 4 inch diameter core was then taken from 830 feet to the base of the hole at 2610 feet.

We were provided with the core index record sheet and a log of fractured, brecciated and incompetent zones by U.S.B.M. During the core inspection at Laramie on 21st January, 1976, we were able to supplement this information from our own observations and also obtain the description of drill cuttings taken from the upper non-cored section of the hole.

Corehole 02-A was cored from the top of the Uinta Formation and the core index record for this hole was obtained during a site visit on 22nd January. The two coreholes are spaced some 610 feet apart.

All of the data retrieved from the two cores is presented in the overall core log in Figure 2. The condition of the cores is expressed in terms of both lost core (%) and a core index CI, the latter being defined by U.S.B.M. as:

$$\text{Core Index (CI)} = \frac{\text{(feet broken core)} + \text{(feet of lost core)} + \frac{1}{4}\text{ no. of joints}}{\text{length of drilled interval}}$$

Core Index is expressed as a percentage, and is in effect similar to the reciprocal of our usually employed RQD i.e., 100% RQD is equivalent to zero CI and represents perfectly competent rock.

The description of drill cuttings indicates that the Uinta Formation is almost entirely siltstone with a varying but minor sandstone content below a depth of 350 feet. Between 720 and 830 feet the siltstone merges with the upper zone of the oil shales. The core logs from 02-A indicate that the Uinta siltstone formation is weak, highly fractured and incompetent. Some 27% of the total cored interval between 200 and 750 feet was not recovered. Between 450 and 650 feet core losses as high as 50% were recorded with correspondingly high

values of Core Index.

The Upper Oil Shale zone, including the Mahogany zone, is generally more competent, but still contains several fractured and collapsed zones. These are particularly severe around 900 feet, 960 feet and over the bottom 50 feet, i.e., 1000-1050 feet of the Mahogany zone. The picture is somewhat different in this region between holes 01-A and 02-A indicating variable conditions even over the 600 feet distance between holes.

The leached zone which extends from about 1050-1400 feet appears to be incompetent overall, with the worst zones around 1100 feet and between 1300 and 1400 feet. The position in 02-A is even worse with considerably less core recovery and correspondingly higher values of core index, averaging 80% between 1200 and 1400 feet.

The Lower (Saline) zone is very competent as illustrated by the high core recovery and low core indices. This zone is characterized by the presence of halite which occurs in two main areas, from 1400 to 1480 feet and 1630 to 1730 feet. The lost core and increase in CI in these regions is presumably due to dissolution of the halite by fresh ground or drilling water. Signs of such action were quite apparent on the core. Also, below about 1500 feet, there are considerable occurrences of Nahcolite both in layer and nodule form.

Below the base of the Parachute Creek oil shales at 2320 feet, the core index and amount of lost core increase. The illite shales are generally weak and difficult to core, tending to part and crumble. However, in-situ they would not be expected to prove a problem for shaft sinking.

Hence, on the evidence of the core inspection and the core logs, we anticipate encountering poor rock conditions between depths of about 200 and 1400 feet, with the exception of a short section within the Mahogany Zone.

4. Borehole Logs

A full suite of borehole geophysical logs were run in 01-A by the Birdwell Division of Seismograph Service Corporation. In Figure 2 we have included "smoothed" versions of these logs by plotting the data, generally, at intervals of 25 feet. Although this loses the fine structure of the original logs it allows us to see the general trends, which is quite sufficient for the present exercise. For a detailed study and correlation, reference should be made to the original logs. Since the diameter of the logging tools was 3-3/8 inches we would expect the borehole fluid to have considerable influence on some of the logs particularly in the upper 800 feet of hole where the drilled diameter was greater than 10 inches.

In this section we discuss each log individually:

(i) Caliper This log reflects the state of the borehole wall i.e. it indicates increases and decreases in borehole diameter in relation to the drilled size. It shows up wide fractures and collapsed zones and is therefore useful for correlation with the core logs. It is also required for compensation purposes, to derive true rock properties such as density and porosity (see later sections), from the recorded logs.

In the Uinta siltstone formation we note major borehole enlargement between 450 and 700 feet depth, corresponding to zones of high core loss and high core index on the 02-A core. Through the Upper, Mahogany and Leached Zones the average recorded diameter is probably between 9 and 10 inches but there are fairly regular enlargements in excess of this, reflecting the incompetent nature of the rock in this area. Since these increases are relatively narrow they do not all show up in our "smoothed" log in Figure 2. Below the base of the Leached Zone there is little borehole enlargement except in the regions of the halite zones where

presumably the presence of fresh water has reproduced considerable erosion of the wall. The generally good state of the borehole below 1400 feet confirms the competent nature of the Lower Oil Shale Zone, as indicated by the core logs. The fact that the borehole is competent below 2300 feet whereas core recovery was poor indicates the presence of weak partings in the Garden Gulch Shales and confirms that this zone should not prove a problem in shaft sinking.

(ii) Temperature The temperature log measures the temperature of the fluid in the borehole as a function of depth, and assuming the fluid has been undisturbed for some time before lowering the tool it should reflect the temperature of the surrounding formation. There will be a steady increase in temperature with depth due to the geothermal gradient which averages 1-2°F per 100 feet increase in depth. Small variations in this gradient will occur due to differences in the thermal conductivity of the various rock types.

In this instance the log shows an overall increase from 90°F at 200 feet to 142°F at 2500 feet i.e. an average increase of $2\frac{1}{4}^{\circ}$F per 100 feet depth.

In Figure 2 we have plotted out the change in temperature at intervals of 25 feet. Thus if the log reflected purely the geothermal gradient we would expect the curve to lie between $+\frac{1}{4}^{\circ}$F and $+\frac{1}{2}^{\circ}$F. The marked discrepancies from such a pattern probably reflects the varying rock characteristics and the presence of ground water. From 200 to about 900 feet the temperature remains fairly constant which could indicate the presence of a generally wet zone. Within the leached zone there are large positive and negative peaks which again probably indicate wet conditions. In the lower zone temperature variation peaks seem to coincide with the halite

zones but below 1800 feet a fairly steady 0.6°F per 25 feet is indicated. The faster than average increase in temperature in the Garden Gulch shales probably reflects the different thermal properties of the formation rather than the presence of water.

(iii) SP (Spontaneous Potential) This is a recording of the potential difference between a fixed surface electrode and a moveable electrode in the borehole. In shaley formations a fairly constant 'shale baseline' is obtained. Opposite permeable formations the curve shows excursions to left or right of baseline depending mainly on the salinity of the interstitial water.

In this log we see a negative peak at the top of the Mahogany Zone (875 feet), which corresponds to poor rock conditions on the core logs and hence probably indicates the presence of ground water. Similarly large negative and positive peaks within the leached zone could indicate permeable zones containing ground water of varying salinity, as indicated during the drilling (see section 5.1).

(iv) Resistivity Three resistivity logs were run simultaneously, namely the 16" normal, the 64" normal and the 18'8" lateral. Since they all indicate the same general trends we have only included one of them, the 64" normal in Figure 2. In the resistivity log, high resistivity zones generally indicate low permeability competent rock and/or the presence of hydrocarbon whereas low resistivity values indicate the presence of saline ground water. Difficulty is encountered when shales are present due to the large volume of bound water.

Through the Evacuation Creek siltstones the resistivity is fairly constant and low reflecting the probable presence of incompetent wet rock. The increases in resistivity within the Upper and Mahogany Zones can be

attributed to the more competent oil bearing low permeability rock. In the leached zone the resistivity again decreases, particularly in the lower half indicating the poor rock conditions and the presence of water. A relative increase through the lower oil shale zone is due to the competent nature of this formation and the presence of oil. The marked decrease in resistivity below 2320 feet indicates the base of the Parachute Creek oil shales and the top of the Garden Gulch shales. The low resistivity here may be due to bound water in these illite shales.

(v) Natural X-ray The gamma ray log is a record of the natural radiation emanating from the formation, and its primary use is as a lithology log differentiating between high activity shales and low activity non-shales.

In this instance we note increased counts at the base of the hole corresponding to the zone of illite shale. Over the upper 2300 feet of the hole we can see lower average counts over the oil bearing and more competent lower, Mahogany and Upper Zones compared with the much less competent leached zone and siltstone formation. In fact the boundaries between the 5 main divisions in the borehole are quite well marked on this log.

(vi) Density The gamma ray density log is unfortunately not compensated for variation in borehole diameter and hence in some zones will not reflect the true rock bulk density. This should be particularly noticeable in areas of large borehole enlargement, as monitored on the caliper log, where the increased volume of borehole water will introduce a misleadingly low value for rock density. In order to obtain an overall qualitative view of this log it should be examined in close association with the caliper log.

We notice that the low 'apparent' densities from 1400 and 1750 feet correspond to the halite and nahcolite zones where considerable dissolution of the borehole wall occurred.

The illite shales at the base of the hole are undoubtedly more dense than the oil shales, where the presence of oil, and in the upper zones water, within the rock matrix will give lower values of bulk density.

(vii) Neutron (Porosity) The same comments apply to this log as to the density log as this is also uncompensated. In general, the neutron log responds to the quantity of water and hydrocarbon present and is used as an indicator of rock porosity.

Again the large count rates in the region of the halite zones reflects the increases in borehole diameter rather than high rock porosity. Generally the neutron log is complimentary to the density log with zones of low density corresponding to zones of high neutron counts (high porosity). This is quite apparent in this instance although the true relevance of the logs cannot be established without including borehole compensation.

(viii) 3D Velocity The basic log as recorded is not particularly useful other than to the experienced log analyst and then only in a qualitative manner. However, when processed together with the caliper, density and neutron logs it provides very useful elastic modulus data as well as providing compensated density and porosity logs.

Unfortunately this processed information was not available to us at the time of preparation of the report, and so we have not included the 3D velocity log in Figure 2.

5. Hydrological Data

The hydrological data made available to us comes from two sources, namely inflow data reported during the drilling of 01-A and pumping tests

conducted in 02-A. Neither set of data is adequate for a detailed design of ground treatment since only very coarse locations of water inflow can be determined. In addition only preliminary results from the pumping tests were available to us at the time of preparation of this report.

(i) Water Data from Drilling Operation The water data recorded during the drilling of hole 01-A is reproduced in Table 1 and illustrated in Figure 2. The data indicates that below 1400 feet, i.e. the base of the leached zone, the formation is likely to be dry. Above 1400 feet water was encountered within the leached zone, and from a depth of 200 feet down to the top of the Mahogany Zone. Although the data for this upper section is very scanty, observations elsewhere in the Piceance Creek Basin (and also as we shall see from the 02-A pumping tests) indicate the presence of two distinct aquifers separated by the Mahogany Zone. Although the most heavily water bearing section of the upper aquifer is likely to be the lower part we must, on the little evidence available, assume this aquifer extends throughout the Uinta formation to the top of the Mahogany Zone.

Due to the unknown effect of the air pressure used during drilling on the hydrostatic head, it is not possible to obtain really precise quantitative estimates for the formation permeability and, hence, the expected inflow to the shaft. However, with a few assumptions we can obtain a range of values.

Assuming the upper aquifer extends from 200 to 900 feet we can deduce a minimum average permeability for this zone of 3×10^{-4} cm/sec (300 millidarcies); when making reasonable allowances for the effect of the drilling air pressure we obtain a maximum average permeability for the zone of 2×10^{-3} cm/sec. (2000 millidarcies).

The lower aquifer is more clearly defined and extends from 1050 feet to 1400 feet. With the same sort of assumptions we estimate a minimum average permeability for this aquifer of 10^{-4} cm/sec (100 millidarcies) and a maximum permeability of some 4×10^{-4} cm/sec (400 millidarcies).

In the actual formation, localized zones of much higher and much lower permeability are likely to exist within each aquifer.

The ground water table was observed at a depth of 115 feet and assuming hydrostatic pressures operate at depth, we estimate the likely inflow to an excavated shaft of 28 feet diameter to be in the range 2000 to 14000 gallons per minute for the upper aquifer and 1200 to 5000 gallons per minute for the lower aquifer.

(ii) Pumping Test Data Two aquifer pumping tests were conducted during the drilling of hole 02-A. Observations of water levels in both 02-A and 01-A were made during the tests. The arrangement in each borehole for the two tests is indicated in Figure 3. A packer was set in the Mahogany Zone in 01-A prior to completion of this hole. A drill pipe passing through this packer made it possible to monitor the water level due to the lower aquifer. The water level in the annulus represented the position in the upper aquifer.

In test No. 1, hole 02-A had been drilled to a depth of 980 feet, i.e. within the Mahogany Zone, and the hole was open between this depth and the bottom of the casing at 190 feet. A submersible pump was set at a depth of 215 feet, and the hole pumped at an average rate of 285 gallons per minute for 15 hours. U.S.B.M. report, on the basis of their preliminary analysis, a transmissivity for the upper aquifer of 2500 ft.^{2}/day. Therefore,

assuming the aquifer extends for 700 feet, this is equivalent to a permeability of 3.6 feet/day or 1.4×10^{-3} cm/sec. In addition, U.S.B.M. state that the drawdown curve is typical of vertically fractured wells, indicating that 02-A may be in the fracture zone of a near vertical fault structure. The data indicated an even larger transmissivity for well 01-A, although no figures are supplied. Hence, anisotropic conditions are present at the site.

No measurable change in water level occurred below the Mahogany Zone in 01-A during the test, indicating that the Mahogany does act as a hydraulic barrier between two aquifers. U.S.B.M. quote a vertical permeability of less than 15 millidarcies (1.5×10^{-5} cm/sec) for the Mahogany Zone. A storage coefficient for the upper aquifer is estimated to be 3×10^{-3} and the aquifer is at least partially confined.

For Test No. 2, well 02-A was cased to 844 feet and a concrete plug subsequently drilled, was set in the hole from this level to a depth of 970 feet. The base of the hole was at 1450 feet, i.e. just into the lower oil shale zone. The same arrangement in hole 01-A, and location of pump, was used as in Test No. 1. In this test the well was only pumped for 15 minutes at a rate of 174 gallons per minute. According to the report "representatives of the Reda Pump Company would not permit the discharge to be valved down enough to allow a longer test". Consequently no water level change was observed in well 01-A.

A transmissivity of not greater than 300 ft.^{2}/day is given for the lower aquifer in the vicinity of 02-A. The fracturing which caused the relatively large transmissivity in the upper part of the section does not apparently persist below the mahogany. However, this may not be valid elsewhere. (In fact, much larger transmissivities have been reported for

the leached zone in other parts of the Piceance Basin (Refs. 1 and 2). Assuming that the lower aquifer extends over a zone of some 350 feet, then its maximum average permeability in the vicinity of 02-A is about 0.9 ft./day or 3.2×10^{-4} cm/sec.

Although it was not possible to compute a storage coefficient for the lower aquifer, due to the lack of response of water level in 01-A, the fact that the water level in the central tube in 01-A was at 125 feet below surface indicates an artesian condition in the lower aquifer.

Therefore, assuming that hydrostatic heads with ground water levels at 115 feet and 126 feet depth act on the upper and lower aquifers respectively, we can estimate likely inflows to a 28 ft. diameter excavated shaft using the permeabilities derived above. For the upper aquifer this gives an inflow of some 9500 gallons per minute in the vicinity of 02-A, but would give more near 01-A, due to the larger permeability. For the lower aquifer, we estimate an inflow of some 3500 gallons per minute.

(iii) Summary Although this form of water test data is not ideal for shaft sinking design studies, it does give some indication of the likely problems, and in this instance gives reasonable consistency between two different sources of data.

The existence of two distinct aquifers is proven, one confined to the 350 feet of leached zone, the other extending over the entire section above a depth of 900 feet. However, in the latter case we can assume by observations of core logs, borehole logs and these water tests that the major problem will exist between 500 and 900 feet depth.

In terms of expected water inflow to the excavated shaft we see that the values derived from the more precise pumping tests fall within the limits of inflows estimated from the drilling data. In overall terms we would therefore expect a total inflow to a 28 foot diameter excavated shaft to be of the

order of 10,000 to 20,000 gallons per minute.

6. Comments on Possible Ground Treatment

The evidence available to date, and described in this report, indicates that conventional large diameter shaft sinking to depths in excess of 2,000 feet, in the Horse Draw Creek region of Piceance Creek, will be difficult due both to very poor rock conditions and the large quantities of ground water likely to be encountered. It would seem essential that some form of ground treatment be applied from ground surface to a depth of at least 1400 feet, i.e. to below the base of the leached zone. Below this depth the lower oil shales appear to be very competent and dry, and should stand well in excavation (3, 4, 5).

Due to the depth of operation and quantity of water involved, pumping methods cannot be envisaged particularly with the added problem of disposal of large volumes of saline water in this locality (6). The only possible means of ground treatment would seem to be freezing or grouting, or possibly a combination of the two. We shall now consider points for and against each method for this situation.

(i) Freezing In ground of such generally poor nature, freezing might seem in the first instance to be the most feasible means of treatment. However, a number of factors must be considered.

Firstly, the ground water flow rate. It is generally considered (7) that a ground water flow rate of some 3-5 ft/day is the maximum that can be tolerated for a successful freeze operation. In this instance we do not have information on hydraulic gradients, and the values of permeability are only very general in nature.

More knowledge of the hydraulic gradients in the vicinity of the proposed shaft and a much finer permeability log of the formation is really

required before we can be certain on whether ground water flow rate is likely to be a problem.

Secondly, ground water composition and temperature will have an effect on the economics of the freezing operation. The presence of salts in the ground water can involve lowering of the required freeze point involving a more extensive and expensive freeze operation. Similarly, the presence of high ground water temperatures would have the same effect. Although at present we do not have a ground water analysis from this site, published data for samples from the leached zone elsewhere in the Piceance (2, 6) indicate that the levels of salts, particularly sodium bicarbonate is not sufficient to depress the freezing point by more than a few degrees, and hence should not prove a problem to conventional brine freezing. However, the presence of extensive halite zones in this particular vicinity might give a different picture. The temperature log for 01-A indicates a formation temperature of some 110°F at a depth of 1500 feet. Such a temperature would probably only cause problems in the time necessary to freeze the formation and possibly the choice of coolant. Note: In a freezing operation at PCA No. 2 shaft in Saskatchewan, it took 95 days for brine temperature to stabilize, when maximum formation temperature was only some 70°F (8).

The third factor of concern is the nature of the rock itself. The problem here revolves around the presence of oil products in the rock matrix and the possible difficulty in freezing such ground. In the kerogen rich parts of the formation, oil yields of up to 50 gallons/ton are predicted, which is equivalent to some 25% by volume. In the worst sections of the leached zone, oil yields are much less, and since the rock is highly broken and water bearing, the presence of small levels of kerogen bound into the marlstone matrix should not cause any problems. In the richer

oil zones of the Mahogany and Upper Zones, where the water content is low, the effect of the freezing is less certain due to the possible creep properties of a frozen oil shale wall. Further study of the mechanical and thermal properties of oil shales (5, 9) are really required before the behavior of the wall could be predicted. Even then, only experts such as Foraky could give a professionally reliable opinion on the security of the ground after freezing, and it is unlikely that they have any experience in freezing oil shales.

(ii) Grouting Undoubtedly a very extensive grouting program will be required to render the entire shaft watertight, as will presumably be necessary if retrieval of the sodium salts is required in addition to the oil shale. On the evidence we have to date we cannot be certain that any zones above 1500 feet will not require grouting. We can, however, expect much of the Mahogany Zone to be relatively dry.

The major problems are undoubtedly the leached zone and the lower part of the upper aquifer where the hydrostatic pressures are high. In the worst 'rubble' zones within these areas it is questionable whether the grouted rock would be competent mechanically. In addition the highly porous zones may make it difficult to contain the grout within the required 'curtain' area, without the injection of indeterminate quantities of grout.

It is possible that a precementation program carried out from the surface, particularly into the leached zone, might reduce considerably the overall water problem encountered when grouting ahead of the shaft. However, the problem of containing the grout would be even more severe in such a process and again indeterminately large quantities of grout might be necessary.

Without a more detailed 'permeability-picture' of the upper 1400 feet of the formation it is impossible to estimate the full extent of grouting operation that might be required.

A comparatively minor but relevant point concerns the level of salts present in the ground water. Special consideration should be made of this in formulating the cement and chemical grouts that would be required.

7. Conclusions

The analysis of three separate forms of data, namely core logs, borehole logs and water tests gives good correlation and confirms the generally reported nature of the formation, and the likely problems for retrieval of the oil shale and salt deposits.

The main hazards facing shaft sinking are two major aquifer zones which are only separated by some 100 feet of the kerogen-rich Mahogany zones:

(i) An upper aquifer zone comprising mainly poor consolidated weak and fractured siltstone merging with fractured Upper Oil Shales over its lower section. The aquifer extend from 200 feet to 900 feet but we can consider the zone from 500-900 feet as being particularly severe with conditions worsening towards the base due to the greater hydrostatic pressure.

(ii) A lower aquifer zone comprising the total length of the Leached Oil Shale Zone and extending from 1050-1400 feet. It is highly fractured and contains frequent collapsed brecciated zones.

The average permeability of the upper aquifer is about 10^{-3} cm/sec in this vicinity and that of the lower aquifer, about 3×10^{-4} cm/sec. However conditions appear to vary considerably over quite small distances, since other data (2) indicates the Leached Zone to be far more permeable than the Upper Zone in other parts of the Piceance Basin. The presence of fracturing near to a fault structure, as apparently detected in the O2-A Water Tests

will contribute to these local variations in permeability. On the basis of the permeabilities quoted above we anticipate water inflows to a 28 feet diameter excavated shaft to be around 10,000 gallons per minute and 3500 gallons per minute for the upper and lower aquifers, respectively.

Below 1400 feet the lower oil shales appear to be very competent and should not provide undue problems for shaft sinking. Fresh ground water in upper and lower aquifers must be kept away from the salt zones which exist almost from the top of the lower oil shale.

To produce a 'dry' shaft during sinking some form of ground treatment will be required over the interval 200 to 1500 feet depth. Of the two processes available i.e. grouting and freezing, we feel, on the evidence available at this time, and subject to certain conditions described in the previous section, that freezing is likely to be the more technically feasible means of ground treatment. Although the water inflow rates are very high they are not too dissimilar to the conditions met in the Nisku formation in Saskatchewan, which was treated successfully by freezing, although ground temperatures appear to be greater here. Assuming the ground water flow rates are not too high we would suggest that the simplest procedure would involve a complete freezing of the formation from the surface down to the competent lower oil shales i.e. to a depth of about 1500 feet.

While not ruling out the possibility of sinking the shaft purely by grouting we do not feel that a sufficiently detailed 'picture' of the formation is available to allow such an extensive program of grouting to be designed, especially since considerable ground strengthening as well as 'water-stopping' will probably be necessary in some zones.

Since neither process, to our knowledge, has been employed on any scale in this formation before, we would recommend that some form of in-situ freezing or grouting trial be attempted prior to a full scale shaft sinking program.

TABLE 1 - WATER DATA AS REPORTED DURING DRILLING OPERATIONS (01-A)

A. First water encountered at 200 feet. After drilling to 471 feet water was entering the well bore at 55-60 gallons per minute.

B. While coring a 7-7/8 inch hole at 915 feet water was reported at the rate of 1000 gallons per minute. Fresh water.

C. At 1025 feet water was 1000 gallons per minute, conductivity 1300.

D. At 1100 feet, water 1000-1100 gallons per minute, fresh.

E. At 1200 feet, water 1375 gallons per minute, conductivity 1350.

F. At 1365 feet, water 1400 gallons per minute, fresh.

G. At 1400 feet, water 1625 gallons per minute, air pressure 450, conductivity 1800.

H. At 1500 feet, water 1625 gallons per minute.

I. Water remained at 1625 gallons per minute throughout remainder of drilling. Conductivity varied from low of 1600 to a high of 2400 during this period. Variation probably due to solution of saline minerals as salt and nahcolite zones were cored.

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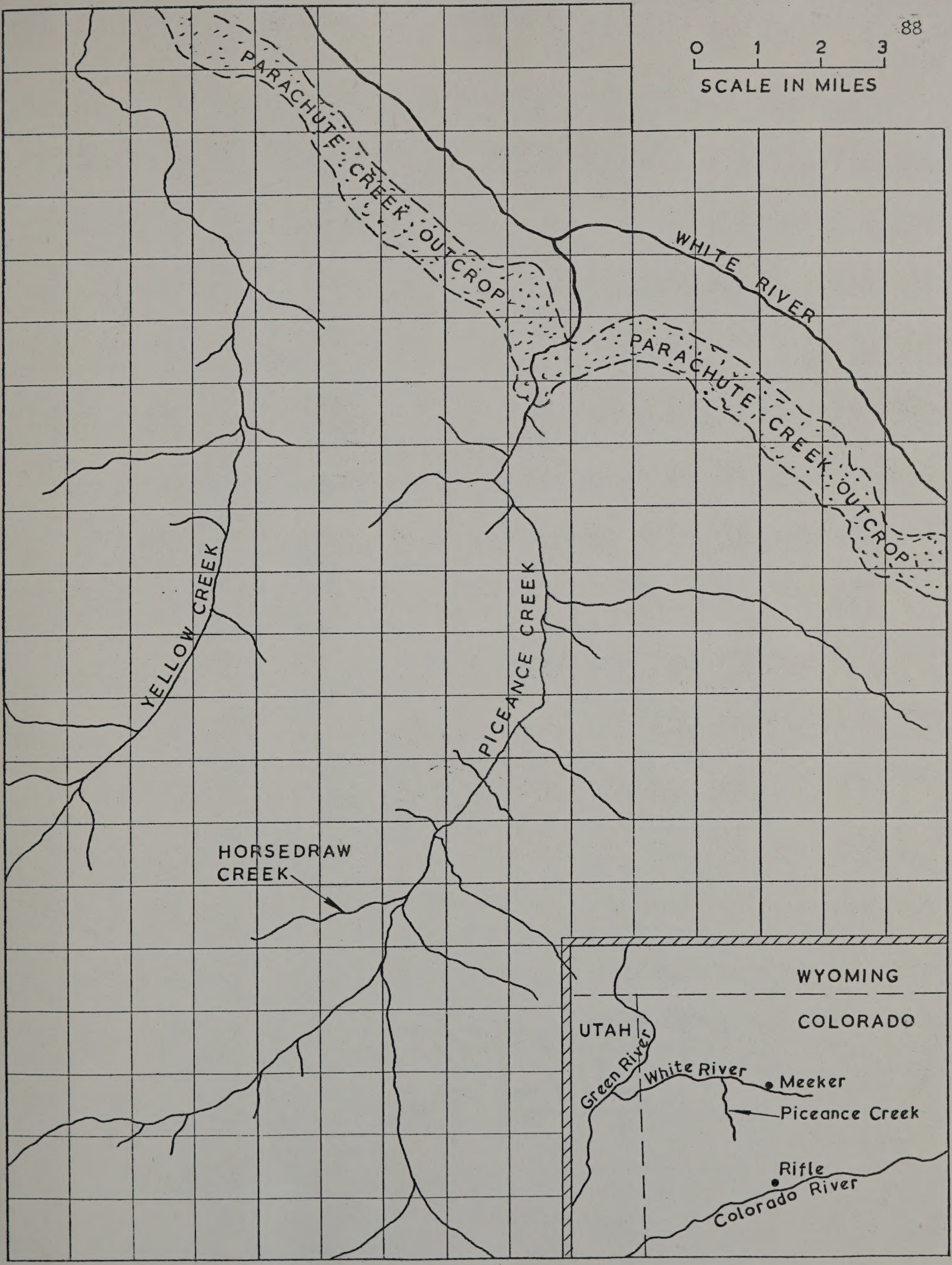
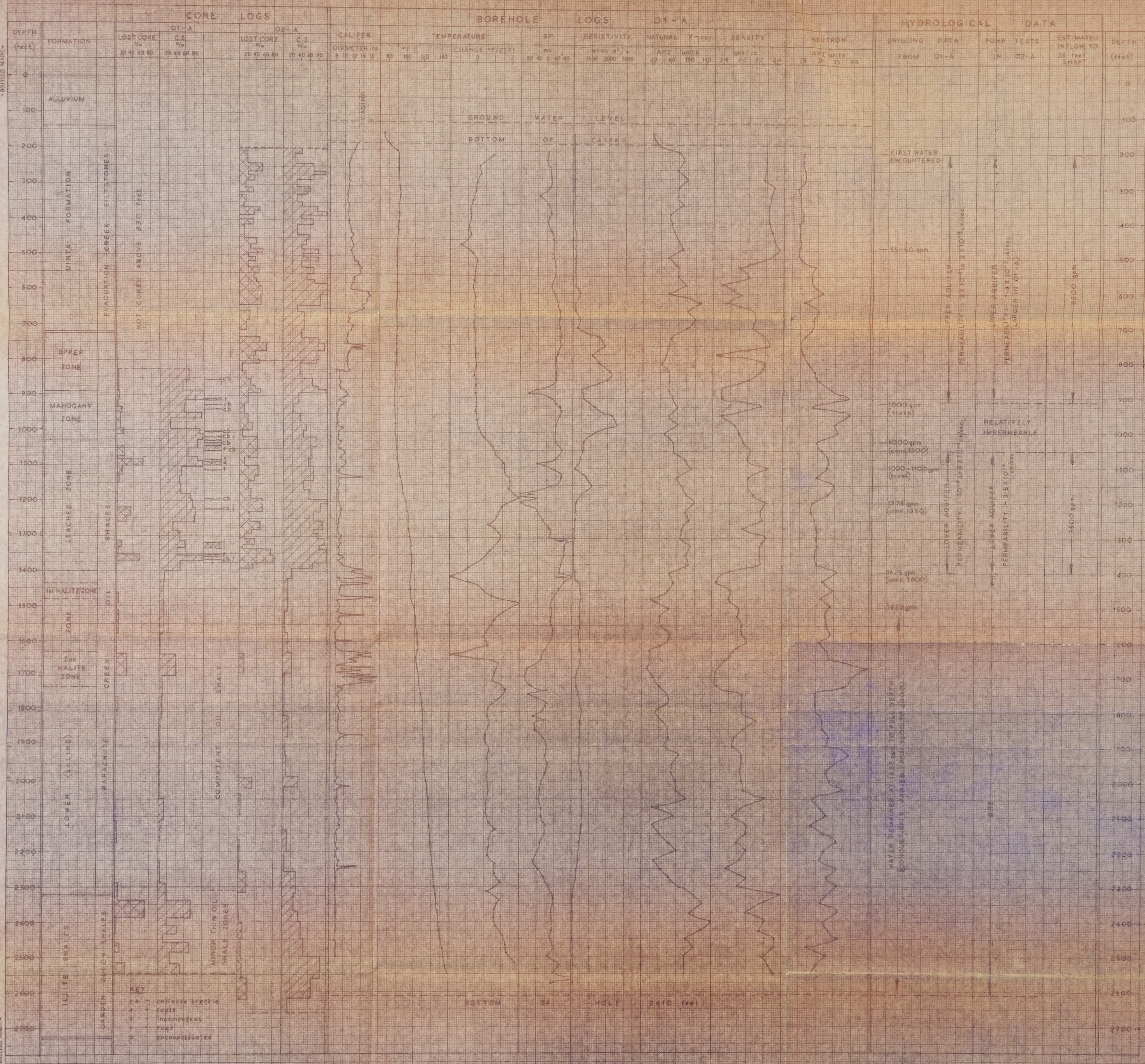
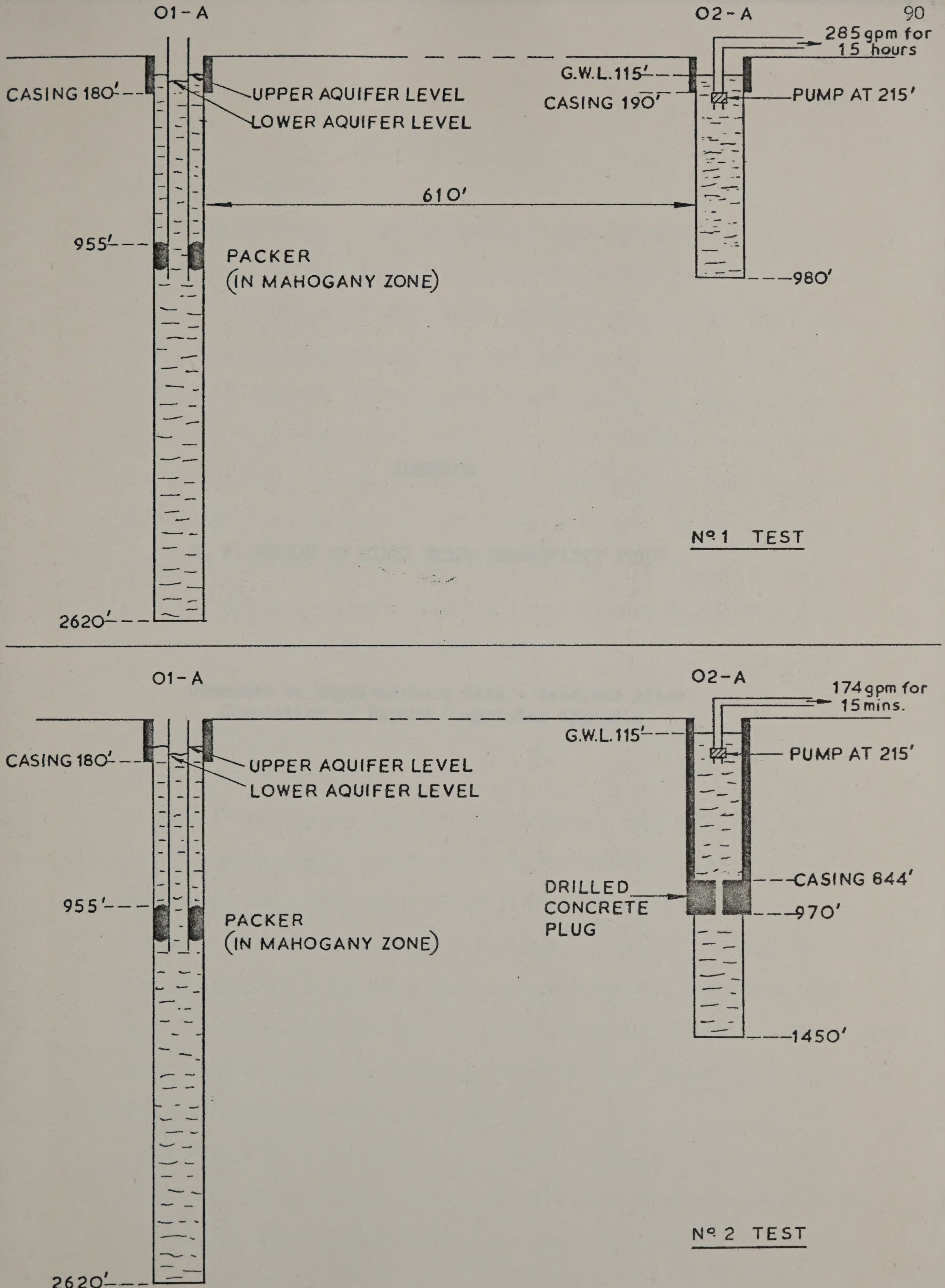


LOCATION OF HORSEDRAW CREEK FIG 1



COMPREHENSIVE LOG OF DATA FROM HORSEDRAW CREEK BOREHOLES O1-A AND O2-A

FIG. 2



ARRANGEMENT FOR PUMPING TESTS IN O2-A

FIG 3

ADDENDUM

U. S. BUREAU OF MINES SHAFT FEASIBILITY STUDY

Comments on Supplementary Data - Received After Completion of Report Comprising Appendix

ADDENDUM

Elastic Properties Log

Elastic properties logs have been received for coreholes 1, 01, 01-A and 02-A. In the case of 02-A, only the Section 300 to 1430 feet is covered. Since our previous analysis was based only on logs from 01-A, we shall only consider the additonal information from this hole.

The data employed on the elastic properties log is computed from the recorded 3D velocity and density logs. Information provided is shear and compressional wave velocities, elastic moduli (Shear, Bulk and Youngs), bulk density, computed porosity and Poisson's Ratio. In the accompanying log we have included this information on a scale identical to that in Figure 2 of the main report. Only the Youngs Modulus is included since the other two moduli follow the same trends.

In this case, the data does not appear to be compensated for borehole enlargement and hence the moduli, density and porosity figures have to be interpreted carefully.

The bulk density and porosity logs correlate very well with high porosity zones generally corresponding to low density zones. The poor conditions of the siltstones below about 500 feet depth is clearly indicated by decreasing density and increased porosity. The high porosities in the Mahogany and lower zones are somewhat misleading since they will include the hydrocarbon content, which in some of the richest oil shale zones is more than 25%. Also, as we saw on the other logs, dissolution in the two halite zones produces anomalies on the logs.

The values of Youngs Modulus in the more competent siltstone zones i.e.$>3 \times 10^6$ psi are fairly typical for this rock type. The lower values within the oil shales reveal their more "elastic" nature due to the high

oil contents. This is also apparent by the high levels of Poisson's Ratio. The abnormally high values for Poisson's Ratio in the lower siltstones are related to the low shear wave velocities which lead to the low values for the elastic moduli in these incompetent rock zones.

Groundwater Analysis

We received analyses of 7 water samples, 2 taken from corehole 01-A and 5 from hole 01, which according to the grid references is some 170 feet from 02-A. We have no details of how the samples were collected or of the size of sample. The samples are defined by a depth range which in all cases is between about 160 feet (bottom of casing) and a particular depth. Presumably samples were recovered during drilling of the holes.

The full analyses are very detailed. In the accompanying table we have abstracted the concentration of the major salts detected in the tests. In 01-A the first test covered the interval 188 to 1313 feet, which includes most of the leached zone, whereas the second test covered the total hole length. The large increase in sodium chloride and sodium bicarbonate concentration in the second test reflects the presence of halite and nahcolite zones in the lower oil shale formation. A similar pattern is observed in hole 01 although in this case the hole was not taken much below the base of the leached zone. It is possible therefore that the large increase in bicarbonate concentration is from the bottom 100 feet or so of leached zone.

The overall level of salt concentration is low compared to those reported by Weichman (1) for a sample from the leached zone in Superior Core Hole 14. In that instance the total dissolved solids was 74000 ppm

i.e. 7.4% w/v. Weichman also states that in parts of the Piceance Basin salinity of the water exceeds 100,000 ppm.

In this case the maximum concentrations are 0.41% w/v sodium chloride and 0.62% w/v sodium bicarbonate. Such levels will not cause a significant depression of freezing point in a ground freezing operation. In the case of grouting some laboratory test work would be essential to obtain the correct grout formulations compatible with this ground water.

Reference

(1) B. E. WEICHMAN — Some effects of the Rio Blanco Project Nuclear Detonation on the Leached Zone in the Parachute Creek Member of the Green River Formation.

Rocky Mountain Association of Geologists 1974 Guidebook, p. 205.

ADDITIONAL DATA FOR FIG 2 OF RESEARCH REPORT R 7/76

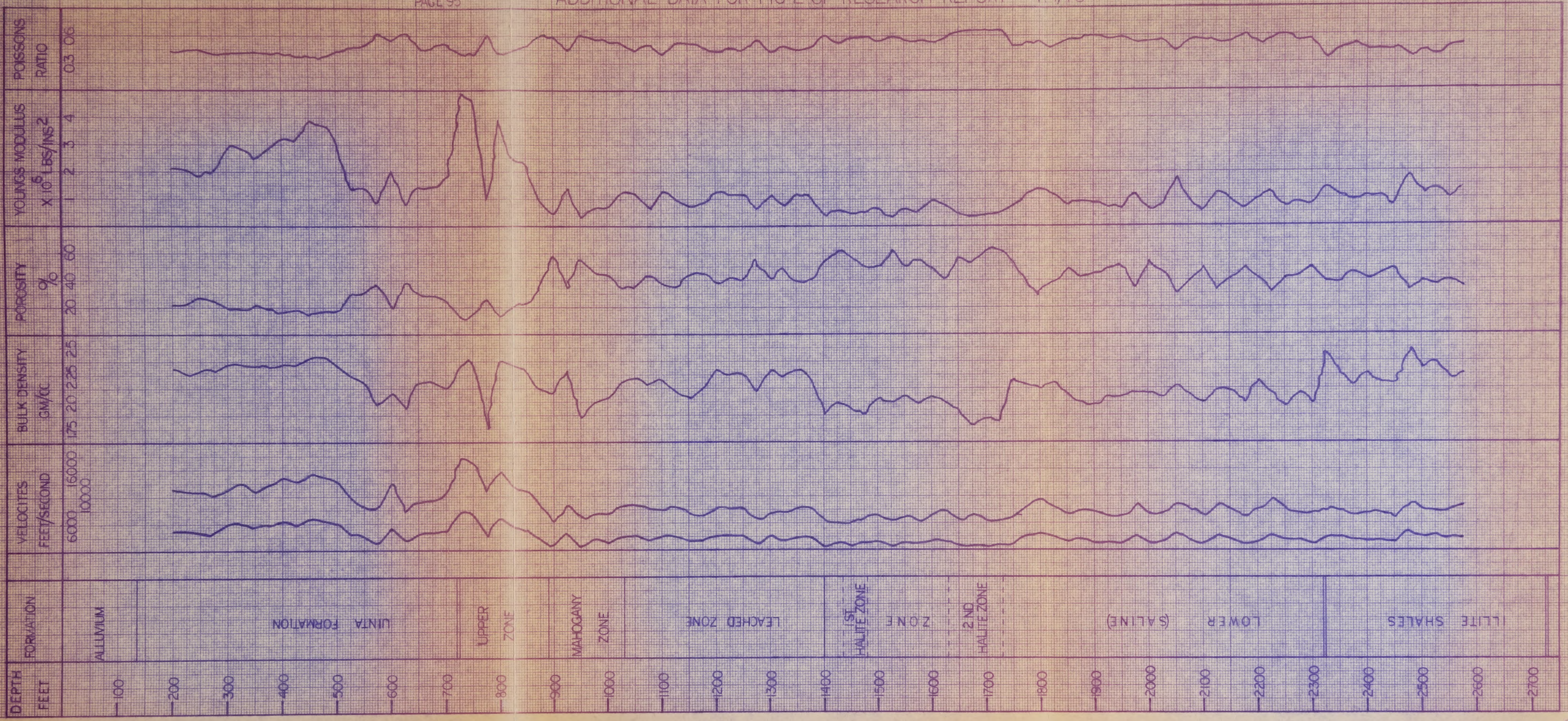


TABLE 2 CHEMICAL ANALYSIS OF WATER FROM BOREHOLES 01-A and 01.

BOREHOLE NUMBER	HOLE 01-A		HOLE 01				
Depth Range of Sample	188'-1313'	188'-2547'	160'-260'	160'-836'	160'-1053'	160'-1207'	160'-1420'
Sodium Bicarbonate, $NaHCO_3$	1074	2685	860	618	676	3305	6183
Sodium Chloride, NaCl	19	4120	-	12	15	117	247
Sodium Sulphate, Na_2SO_4	117	140	-	265	287	154	31
Magnesium Sulphate, $MgSO_4$	95	25	665	240	195	180	170
Calcium Sulphate, $CaSO_4$	48	20	240	71	58	56	51
Total Dissolved Solids	1350	6980	1765	1200	1236	3900	6837
pH	8.1	8.1	8.4	8.3	8.1	8.1	8.3
Temp. °C	18	18	13.5	19	18	20	20
Specific Conductance	1400	10400	1900	1300	1375	3500	5900

Note: Salt concentrations are in milligrams/litre i.e. ppm.

U.S.B.M. Piceance Creek Shaft Study - Phase II

Appendix IV

Employee Classifications

This appendix shows the employee classifications and schedule of wage rates used in estimating the construction costs and the maximum numbers employed in each classification.

Engineers	Project Manager
	Project Engineer
	Shaft Engineer
Clerical	Accountant
	Clerk
	Warehouse Attendant
Supervision	Superintendent
	Walker
	Mechanical Superintendent
	Lead Miner
Trades	Mechanic
	Lead Mechanic
	Electrician
	Carpenter
Operators	Mucking Unit Operator
	Hoistmen
	Front-end Loader Operator
	Batch Plant Operator
Miners	Miners
	Raise Miners
Laborers	Toplander
	Powderman
	Laborer
	Dryhouse Attendant

Schedule of Wage Rates for July 1978 to July 1979

U.S.B.M. Piceance Creek Shaft Feasibility Study

Job Classification	Rate ($ per Hour)
Shift Boss	10.63
Miner	9.69
Toplander	9.10
Carpenter	9.69
Powderman	9.21
Laborer (Surface)	8.78
Truck Driver	8.78
Electrical Repairman	10.40
Mechanic	10.40
Hoistman	10.40
Mucker Operator	10.40
Front-end Loader Operator	10.40
Batch Plant Operator	10.40

U.S.B.M. Piceance Creek Shaft Study

Appendix V

Camp Costs

The costs are based on a construction-type camp facility being provided in an area which will give ready access to the construction sites. Trailer-type accommodation and messing facilities would be provided. However, it was not intended, and no provision was made, to accommodate families.

In estimating the size of the camp it was assumed that thirty-five persons from the labor force during construction would not need camp accommodation. These people would either be local to the area, or, if from the outside, would rent accommodations or take up lodgings in some of the local villages.

The three main component parts of the total camp costs as given in the summary of estimates are:-

Capital cost of trailers, furnishings and installation costs. (These costs written off in total.)	$ 250,000
Cost of kitchen, cleaning and administrative staff	$ 185,000
Cost of food subsidy at rate of $49.00/week/man	$ 210,000
	$ 645,000